HORIZON MINING

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FOREWORD

By E. H. Browne
Director-General of Production, National Coal Board

It is well known that there are fundamental differences in the basic system of layout of British and Continental mines. The cause of this lies in the inter-related effects of different geological conditions and history. The coal reserves of Britain, though rich, are spread over wide areas and the quantity of workable coal in the strata is generally less than on the Continent, particularly in Germany. In the Ruhr, for example, the 'density' is about four times as great as the average for Britain.

Our coal measures are often faulted and inclined, but they are not folded to the extent of the Ruhr; and in most of our coal fields the seams outcrop, whereas on the Continent they are often overlaid by thick and heavily water-bearing strata. In the Ruhr, shaft sinking is difficult and expensive and concentrated outputs are necessary to pay for it; and the average depth of winding is about double that of Britain.

There are other dissimilarities, though they must not be over-generalised, which have combined to cause the differences in mining systems. In this country the coal has usually been worked first by shallow shafts and drifts near the outcrop, with deeper shafts as time has gone on. Transport in the main has been in roadways in the seams which have been worked individually. On the Continent the horizon-mining system was in most cases a necessity from the start.

It is generally recognised that in the situation today, the horizon-mining system, or a modification of it, is often necessary for the economical exploitation of our deeper coals. Because of the greatly varying conditions in the different coal fields of the country, the extent to which this change in technique is necessary varies considerably. Some of the schemes which the Board have in hand, and others which are planned, are on classical lines of Continental horizon mining; others are a compromise between traditional British methods and those of the Continent; and in some cases the horizon system would not be appropriate at all. With the many instances in which Continental practice is to be adopted partly or wholly, there is a

pressing need for concise and easily accessible information on the subject. The Board have therefore been very fortunate in obtaining the services of such an eminent author and teacher as Professor Fritzsche, of Aachen, to undertake the difficult task of writing, as it were, a textbook on horizon mining designed for use by the British mining engineer. Professor Potts has been kind enough to collaborate with Professor Fritzsche in order to provide the essential background of British conditions. They have, of course, been allowed freedom in its compilation.

I am sure that this book will be of real value at this critical time in the history of coal-mining in this country, when reconstruction and development on a quite unprecedented scale is being conducted; and that the mining engineers of the country will make good use of it.

PREFACE

The National Coal Board invited the authors to collaborate on a book dealing essentially with German mining practice in the layout and development of the Horizon-mining System. The Board had organised numerous visits of British mining engineers and students to the Ruhr and it was thought that a book covering the wider aspects of horizon-mining development would be of great assistance in providing a detailed picture of a system, for the introduction of which several of the British coal fields offered considerable scope.

The National Coal Board have assisted very greatly in its production and in particular the authors have had the benefit of the advice and wide experience of Ir. J. Bakker, the Board's adviser on Continental mining methods, during the preparation of the text.

While it should be appreciated that there have been many technical developments in numerous aspects since the commencement of writing the book, the basic principles of horizon-mining development remain unaltered. The authors have attempted, with the aid of profuse illustration, to show the winning out of a roadway network on the horizon-mining system, and to restrict the book to this development and the associated haulage systems without reference to actual winning methods at the coal face. In many cases the equipment described is of German origin, although British equipment and practice have been included in those sections where such reference is more relevant and it was considered that British conditions and experience should also be recorded.

In order to illustrate the general introduction of horizon-mining projects in the British coal fields, the National Coal Board have assisted the authors in the preparation of the Appendix, which includes brief descriptions of several of the projects initiated by the Board and now in the course of development.

The authors have not attempted to be authoritative on the many other aspects associated with any mining development, but have stressed the important factors which have to be considered where the horizon-mining system of development is introduced.

The book has drawn information from many sources, both in Germany and in this country, and the authors take this opportunity

to thank all those who have assisted in any way in its preparation. The many manufacturers of mining equipment in Germany and Britain referred to in the text have been most helpful and generous in providing illustrations where they were required.

Professor Fritzsche wishes to thank Dr. A. Hohenstein for preparing many diagrams and Mr. P. H. Oidtmann for his help in the revision of the proofs, while Professor Potts would like to express his gratitude to Mr. A. C. Talbott of the Engineering Department, King's College, for reading the text and Mr. D. Dodds, draughtsman in the same Department, who has made so many excellent illustrations.

The help is acknowledged of the following manufacturers in providing illustrations of their mining equipment which are reproduced in the book:

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CONTENTS

Part III. Methods of Working Designed to Reduce or Prevent Surface Damage and at the same time Provide Suitable Strata Control Underground

CHAPTER 3. DEVELOPMENT IN STONE

Part I. Driving Level Cross-measure Drifts, Lateral and Inclined Drifts

Part II. Support of Main Roadways

Part III. The Construction of Large Chambers Underground

Part IV. Sinking and Driving of Staple Shafts

CHAPTER 4. DEVELOPMENT IN COAL

Part I. The Development of Gate Roads

Part II. Methods of Support in Gate Roads

Part III. Driving Rises or Inclined Roads in Coal

CHAPTER 5. TRANSPORT

Part I. Transport in Gate Roads

Part II. Transport in Staple Shafts

Part III. Transport in Level Main Roads

CONTENTS

CHAPTER 6. VENTILATION

CHAPTER I

THE GENERAL PRINCIPLES OF THE HORIZON MINING SYSTEM

PART I

THE EVOLUTION AND APPLICATION OF THE SYSTEM

Section 1. *General Introduction*

In the development of every mine, whether it be a coal or metalliferous mine, a grid of main and secondary roadways is set out to serve the 'winning' points and to connect these working places with the outlet to the surface, which may be either a shaft or a surface drift. The extraction of the mineral at the working face is conducted by hand-got methods or may be highly mechanised. In the case of coal-mining, the sequence of operations may include the complete mechanisation of the coal-getting and loading operations using simultaneous cutting and loading machines. The network of roadways divides the area to be developed into sections of suitable dimensions for extraction. The roadways are designed to serve the following purposes:

(*a*) The transportation of the mineral from the face to the shaft bottom or commencement of the surface drift.

(*b*) The transportation of materials and supplies.

(*c*) The transportation of men.

(*d*) For purposes of ventilation.

(*e*) For conducting the power supply (electricity and/or compressed air) to serve the underground system.

(*f*) To serve a drainage system.

The layout of the road network will be mainly governed by the consideration of the geological conditions, which will include:

(*a*) The dip and number of seams and the distance between them.

(*b*) The presence of major faults or dykes.

In an area in which the coal seams are fairly flat, the main and secondary roads are driven in the seam as in conventional British and American practice. All transportation then takes place at the same level or 'horizon' which is in the seam itself. The secondary and main

transport roads and ventilation airways are all in the seam and at the same horizon. This implies that every seam being worked within the boundaries of development must have an equivalent and separate roadway network. Generally, development in depth is such that not more than two seams are being extracted within the same immediate area and, invariably, extraction is limited to the upper seam and followed at a later date in the lower seam or seams.

The term 'level', in Continental practice, covers all main and secondary roads driven at the same depth or in the same horizon. According to this definition, therefore, in the conventional system every seam represents an individual level, and the number of levels developed is equal to the number of seams being worked. Since each seam is worked at a single level and the layout of the roadway network relates only to a single seam, the conventional system could be termed 'single level mining'.

With increasing dip, it becomes necessary to drive some of the main roads to the dip of the seam, which roadways, if extended to a considerable length, make economic transport, ventilation and drainage difficult to maintain.

In highly inclined conditions, another system has been employed to divide the mining property into development areas. A system of horizontal or level main roadways is driven; two such systems, at least, being necessary for the full development of the area. The main roadway network shows a marked difference from that of the 'single level' system. The system used in these conditions incorporates the following features:

(*a*) The main roads are driven on a level gradient, independent of the dip of the seams or the variation of their dip.

(*b*) These roads are almost exclusively driven in the rock, running perpendicular, or parallel (or in some cases in a direction between these extremes), to the strike of the seams. The roadways at right angles to the strike, or diagonal to the strike direction, are called 'cross-cuts' or 'cross-measure drifts', and are always driven in the rock. The roads parallel to the strike are 'lateral roads', which are usually driven in the rock but, in some cases, may be driven in the seam.

(*c*) An important distinction from the 'single level' system is that the main horizontal roads do not handle the complete output from one seam only, but from all seams intersected or lying between a pair

of cross-measure drifts which have been driven at two different depths or horizons one above the other.

(*d*) Unlike the 'single level' system, in which the ventilation and the transport of coal, men and materials are conducted at one level only, this method of development has the ventilation which is normally 'exhausting', but can be 'forcing' arranged on an ascensional system, with the lower horizon as the intake and the upper horizon as the return. The reverse is the case in a descensional system, when the lower horizon is the return and the upper horizon the intake; whereas the transportation of coal is normally confined to the lower horizon, but materials and supplies may be transported in both horizons.

(*e*) Generally, roadways driven in the seam, either on the strike or on the dip, are secondary roads such as gate roads, mothergates, tailgates or centregates.

Since the main roads are horizontal and are independent of the inclination of the seam or seams, the system which includes more than one level is termed 'horizon mining'. This must imply that the main roads are driven in the rock and that a system of separate levels is introduced, in which the intake and return air travel at different horizons.

Frequent reference is made in the following parts of this book to strata inclination, and the inclination is assumed to cover the following range:

Flat measures	-20°
Semi-steep measures . .	$+20^\circ-40^\circ$
Steep measures . . .	$+40^\circ$

Whereas the previous description has been applied to working in highly inclined measures, the same system can be applied to seams of less inclination and in flat or level seams. In these circumstances two levels are again driven, the lower level being the main haulage level and intake airway, while the upper level is the return airway. The same network of cross-measure and lateral drifts is required in both horizons as in the case of the steeper seam layout. In the same manner as in the development between horizons in steep seam conditions, the horizontal roads driven provide for the transport of coal, men and materials for more than one seam. The number of seams served by two horizons will vary according to conditions, and may be between

two and ten but is usually three, four or five in flat formations.

A connection between horizons is required in order to facilitate ventilation and the transport of coal, men and materials. This connection is provided by staple shafts driven either immediately between each horizon or between the intervening seams and the upper or lower horizons. Thus a staple shaft may connect level to level direct, or connect the gate roads along which the coal is transported from the face with either the upper or lower level. In some cases inclined drifts in stone or coal replace the staple shafts; in the cases of inclines in coal, this is only resorted to if the inclination is suitable, which is usually governed by the system of coal transport being employed.

In the case of staple shafts sunk from level to level direct, these give a ventilation connection to the intervening seams from the main intake or haulage level. If sunk for the transport of coal and as part of the intake ventilation system, they connect gate roads and the main haulage level. If they are sunk from the return or upper level to the seams, they serve as return airways and for the transportation of men and materials from the upper level to the face.

Since the network of main roads is horizontal and thus independent of the dip or undulation of the seams, locomotive transport, which is undoubtedly the most efficient and economic main transport system, can be introduced. It is still possible to introduce locomotive haulage in a 'single level system' if the main haulage roads are driven horizontal, either in, immediately above or below the seam level, thus avoiding differences in the gradient of the seam. These roads would be suitable for locomotive haulage and serve as main haulage roads for at least some districts of the mine.

Thus, the horizon mining system, in the wider sense of the word, can be described as a system of development which allows locomotive transport to be adopted as a means of main haulage.

In the case of a layout in which only one level is used, the term 'semi-horizon mining' could be applied, the absence of a separate return ventilation level and staple shaft connection between upper and lower levels being the main differences from normal horizon mining practice.

Section 2. The Evolution and Application of the System

The horizon mining system was first introduced in metalliferous

mining, which was practised many years before coal-mining commenced. The development of steeply inclined veins can only be carried out by working from the surface and developing in depth down the hade of the vein. The limit of development in depth of any section of the vein is governed by the method of extraction employed. The general development system is to 'block out' an inclined pillar, bounded by inclined shafts or 'rises' in the vein as the transverse limit and by upper and lower roadways driven along the strike, these roadways being 'levels'. The distance between these levels is governed by the facility to 'draw' broken ore from the inclined pillar or 'stope' to the lower haulage level and by the ease with which waste filling can be introduced from the upper level via the rises at each end. In normal practice the distance between levels varies from 120 to 200

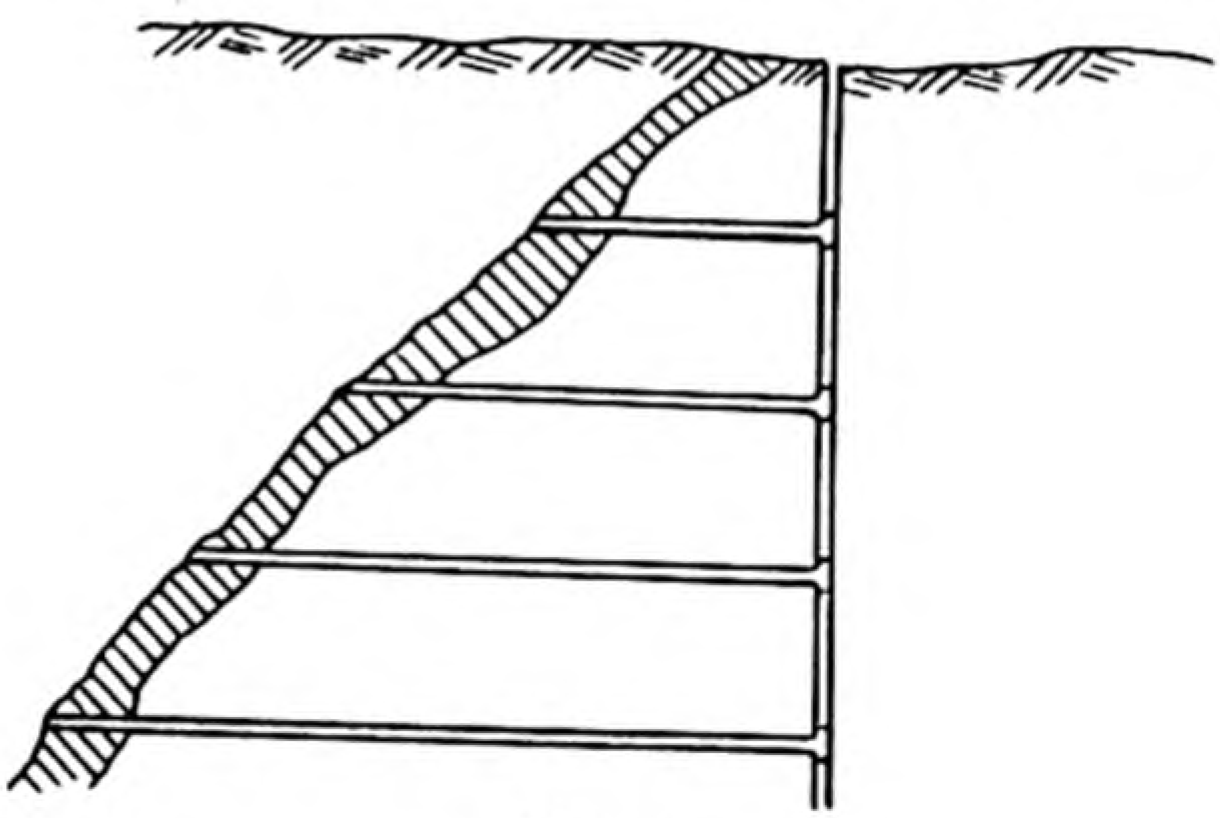


FIG. 1

feet. Thus, in the course of working in depth, a system of horizontal levels is developed 120–200 feet apart, along which ore and supplies are transported. Each network of roadways driven at the same level, consisting of roads in the vein, cross-measure drifts driven from the shaft to intersect the vein, or cross-measure drifts from the surface, represent one horizon or level. Fig. 1 shows an example of working in a lode or vein. The same system is applied to any deposit of considerable extension in depth when a system of horizon mining is the accepted method of development. Fig. 2 shows the extraction of Blue ground (diamond-bearing clays) by means of horizontal levels in Kimberley, South Africa. The deposit has been divided into vertical sections by horizontal grids or networks of roadways, each horizon representing one level. Every section is limited by one upper and lower level, extraction taking place between these levels. The ore is

hauled along the level at each horizon and raised to the surface through the shaft. Fig. 3 is a further example of horizon mining practice in metalliferous mining.

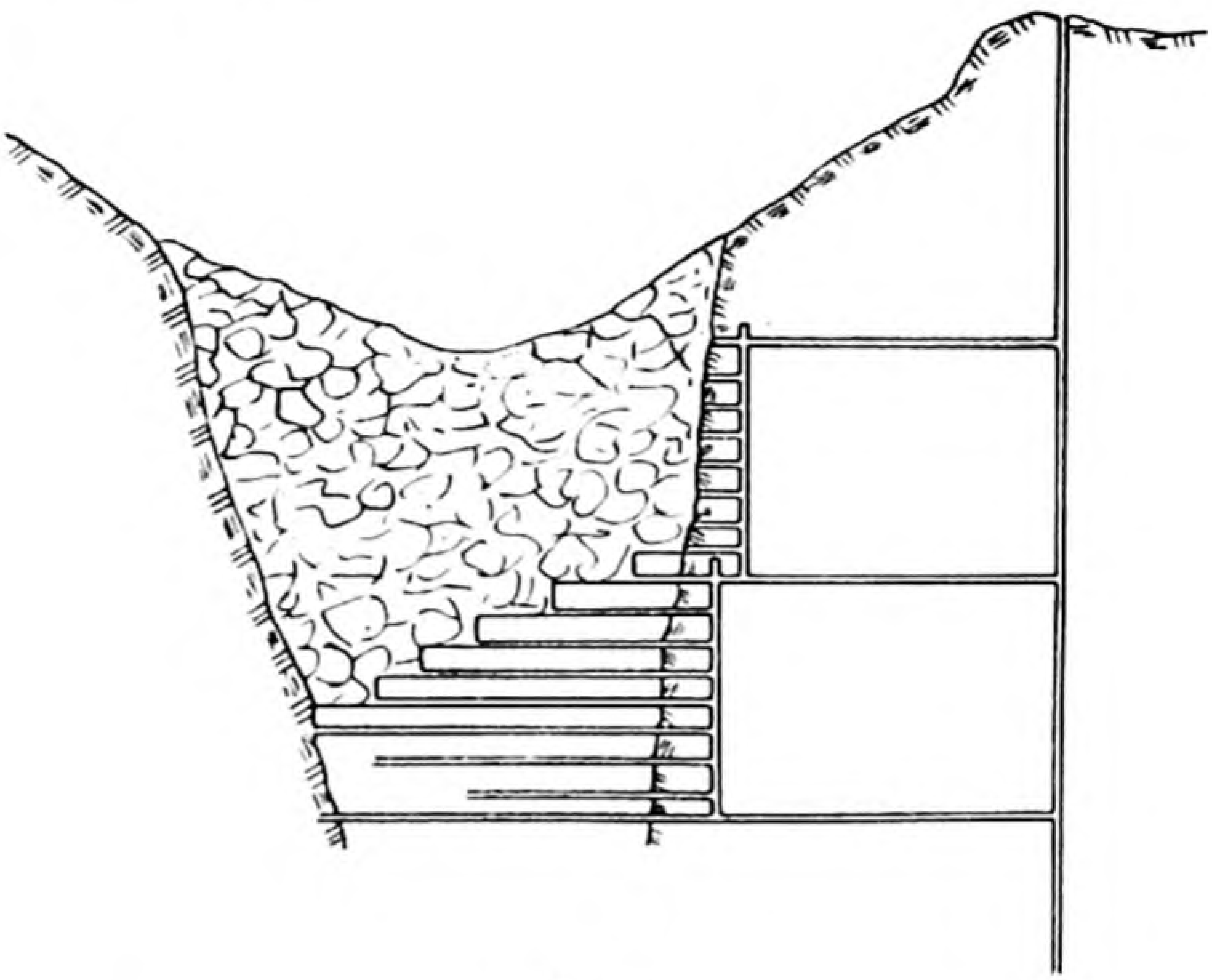


Fig. 2

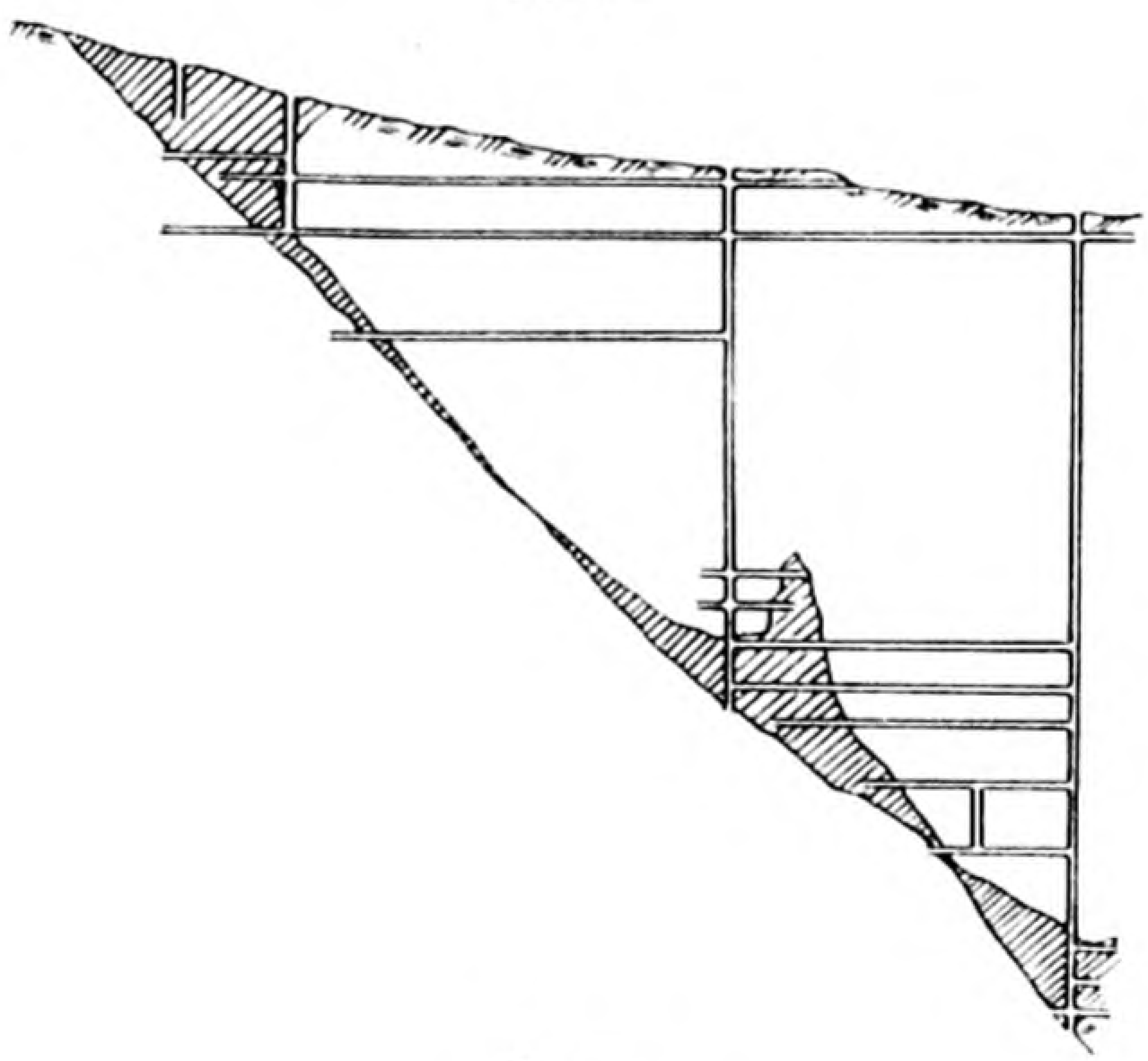


Fig. 3

Coal-mining in the European continent began in the valleys near Aachen, in the Ruhr, in Southern Belgium and in France. The first seams encountered were in areas of folded formation in the coal-

measure strata and had a semi-steep inclination. The development of these seams provided the same problems as have already been described in metalliferous mining development. The seams were generally developed through adits, or surface drifts, driven from the seam outcrops in the valley. As deeper seams were developed, shafts were sunk and the seams which did not outcrop were intersected with cross-measure drifts driven from the adits or shafts. Eventually the first horizontal network of roadways, representing an horizon or level, was developed, this first horizon serving for the extraction of the coal lying between the level and the surface. Further extension in depth established the horizon system of development. Fig. 4 illustrates the development of the horizon system in the early days of mining in the Ruhr.

Fig. 5 shows an interesting example of horizon development in an

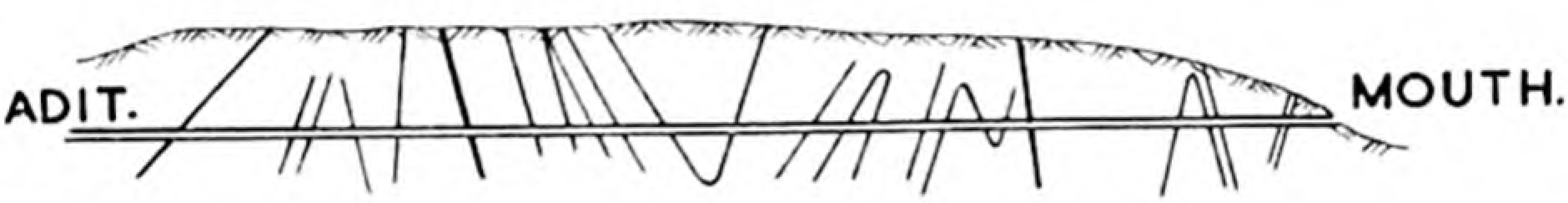


Fig. 4

English coal field, in which the lower haulage horizon was a canal or waterway system, and barges acted as the main transport medium.

Further development of the coal fields on the Continent introduced variable conditions of seam inclination and flat seams were encountered. In the development of such seams, roads were driven in the coal as far as was practicable, driving in the seam being cheaper than in the rock; thus, the conditions were similar to the conventional system encountered in Great Britain and the U.S.A. The horizon system of mining was introduced in mines working the steeper seams. The increase in the output, connected with greater difficulties in haulage and ventilation, made it advisable to adopt the same system for flat measures, horizontal roads being more suitable for efficient haulage. They could also be used to transport the coal from several seams which were being worked simultaneously. Having intake and return airways at different levels is of considerable advantage for efficient ventilation.

It is easily understood that, due to this experience, the predominant system practised on the European continent is the horizon mining system. The system has, in fact, been applied to the more

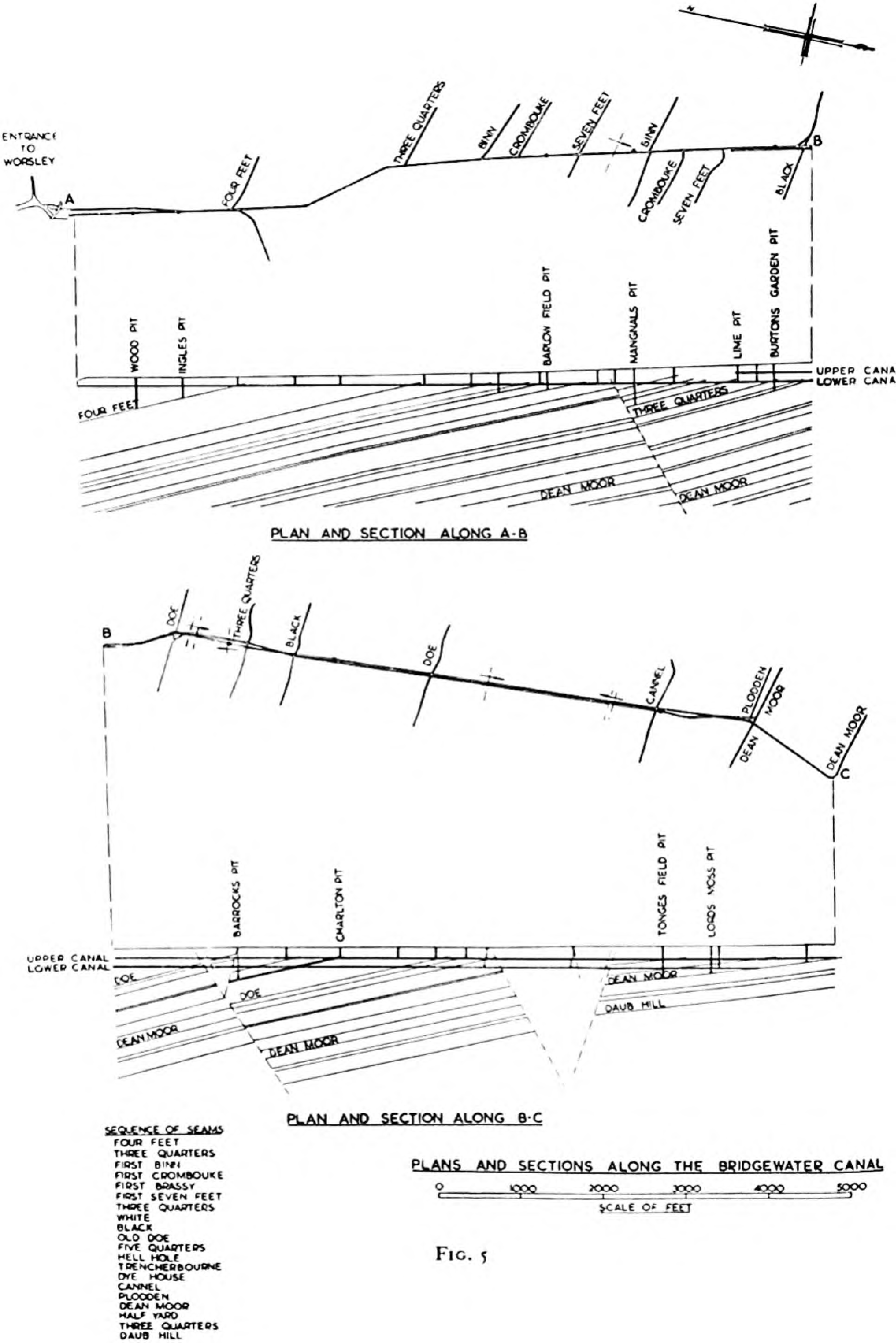

ENTRANCE TO WORSLEY
A
B
FOUR FEET
THREE QUARTERS
BINN
CROMBOUKE
SEVEN FEET
BINN
CROMBOUKE
SEVEN FEET
BLACK
WOOD PIT
INGLES PIT
BARLOW FIELD PIT
MANGNALS PIT
LIME PIT
BURTONS GARDEN PIT
UPPER CANAL
LOWER CANAL
FOUR FEET
THREE QUARTERS
DEAN MOOR
DEAN MOOR
PLAN AND SECTION ALONG A-B
B
DOE
THREE QUARTERS
BLACK
DOE
CANNEL
PLODDEN
MOOR
DEAN
DEAN MOOR
C
BARROCKS PIT
CHARLTON PIT
TONGES FIELD PIT
LORDS MOSS PIT
UPPER CANAL
LOWER CANAL
DOE
DOE
DEAN MOOR
DEAN MOOR
DEAN MOOR
DAUB HILL
PLAN AND SECTION ALONG B-C
SEQUENCE OF SEAMS
FOUR FEET
THREE QUARTERS
FIRST BINN
FIRST CROMBOUKE
FIRST BRASSY
FIRST SEVEN FEET
THREE QUARTERS
WHITE
BLACK
OLD DOE
FIVE QUARTERS
HELL HOLE
TRENCHERBOURNE
DYE HOUSE
CANNEL
PLODDEN
DEAN MOOR
HALF YARD
THREE QUARTERS
DAUB HILL
PLANS AND SECTIONS ALONG THE BRIDGEWATER CANAL
0
1000
2000
3000
4000
5000
SCALE OF FEET

FIG. 5

recently developed coal fields exploited during the last thirty to forty years, particularly in North-east Belgium and in Holland. In these new coal fields the measures generally are not highly inclined and modern mines have been developed in the last twenty to thirty years with a daily production of 4,000–10,000 tons per day.

The horizon system of mining has also been introduced for the working of flat seams in Western Germany, France, Russia, Poland and Czechoslovakia. It should be emphasised that this system has been applied with success to those mines working flat seams where the haulage of large tonnages of coal to one point, the shaft bottom, is both technically and economically most efficiently conducted by the horizon mining system.

There are cases where the 'single level' system is employed on the Continent, in some of the potash and in all the Brown coal mines in Central Germany, where flat strata formations exist. The roof or upper immediate strata in the Brown coal area consists of sand and gravel, drivage in which would entail many difficulties.

The application of the 'single level' system, which is normal to British mines, is understandable since the coal-measure strata is usually of low inclination and in the past extraction has been confined to the best seams. The distance between the better seams is usually high, and the application of level horizons in the intermediate strata would have been expensive. Similar conditions exist in coal-mining areas in the U.S.A. Seams of lower quality and those considered uneconomic to work are now being developed in Great Britain, due to the advance in coal-face mechanisation and coal-cleaning technique. Rope haulage, which has been used to a great extent, is considered to be costly, while the advantages of locomotive haulage are becoming more and more apparent. Many mines are therefore being reorganised to change over to the normal horizon mining system or to a combination of horizon mining and the conventional dip-mining methods, in which horizontal roadways are being driven to accommodate locomotive haulage.

PART II

THE PRINCIPLES OF DEVELOPMENT OF THE NORMAL HORIZON MINING SYSTEM

Section 1. *General Introduction*

The predominant feature of normal horizon mining in coal mines is the driving of horizontal main haulage roads, comprising the main haulage level. The roads are, as a rule, below the seams to be worked. Above the haulage level and the seams to be developed, another horizontal network of roads is driven, comprising the ventilation level; thus, at least two levels are required to deal with the extraction of the intermediate coal reserves.

Before development can commence in depth, and before the present reserves are extracted completely, a deeper level must be driven in advance to take the place of the existing haulage level. Ultimately, the existing haulage level will become the ventilation level, when the lower series of seams comprising the new reserves are being worked. Thus, the life of the main roads of a level is dependent upon the time required to extract the reserves above and below that level, and between the limits of the three levels in use, of which the level concerned is the intermediate one.

It is possible that, when the lower level is in use as a main haulage level, two of the upper and previous haulage levels are still in use as ventilation levels. In the case of the upper level, probably only parts of the level would be used.

Two main haulage levels may be in use simultaneously, in which case the life of each level is then longer than if a single haulage level is used, providing that greater coal reserves are developed above these levels. It is considered, however, that the best practice is to have one main haulage level to which all extraction is concentrated, and men and machines are working at their highest capacity. Apart from this consideration, haulage on the upper level will be affected by the extraction taking place below that level. Such extraction may result in changes of gradient taking place on the upper haulage level and maintenance of the roadways would then become expensive. The presence of possible fractures may provide access for methane from the lower measures into the upper haulage level, with consequent difficulty in maintaining an adequate ventila-

tion in a roadway where trolley and diesel locomotives may be in use.

Exceptions from this rule may be justified by geological conditions, or reasons of selective mining of seams of different quality at different depths. It may be necessary to wind coals of different types from several levels at the same time, in which case two haulage levels are an advantage. It must also be considered that the change-over from an upper to a lower haulage level takes place gradually. For a certain period, which may be months or even years, an increasing percentage of the total output will be wound from the lower level, while the percentage obtained from the higher level decreases. This schedule of extraction is assisted if the main winding shaft is fitted with four cages, two of which wind from the lower level and two from the upper level. An intermediate stage of this development may also be arranged where a staple shaft provides an outlet from the lower level, which is in course of development, to the intermediate or main haulage level, the main shaft winding being still confined to the upper main haulage levels.

Section 2. Development of a Main Haulage Horizon

(*a*) *Classification of roads.* The network of roads representing the main haulage level has to handle all transport and ventilation as well as provide access to the intermediate seams; they also provide points of attack for developing the reserves into suitable panels or units. For this purpose roads are driven at right angles to the strike (cross-measure drifts, cross-cuts, cruts, slants, etc.) as well as parallel to the strike (lateral roads, lateral drifts or laterals). The gate roads, comprising mothergates, centregates and tailgates, are horizontal roads driven in the seam, handling the transport of coal, men, waste and supplies, and they may be at the level of the haulage horizon, or at an intermediate or sub-level. Where gate roads are driven along the dip of the seam, these roads are termed 'rises', as in metalliferous mining practice.

(*b*) *Cross-measure drifts.* Assuming the development of a series of seams having a slight inclination, some or all of these seams will intersect the main haulage level. Fig. 6 is a section showing the seams and the relative position of the main levels. Fig. 7 shows the lines of 'cut-off' of the seams with the level.

Cross-cuts are driven, intersecting the seams and dividing the

reserves into districts of a suitable size. The cross-cut beginning at the shaft bottom is generally called the main cross-cut and is, in fact, a 'level extension' of the vertical winding shaft; this cross-cut may be driven to one or both sides of the shaft position, as in Fig. 7. A single cross-cut will only serve a limited extension of development on the strike, and normally several cross-cuts are required to divide

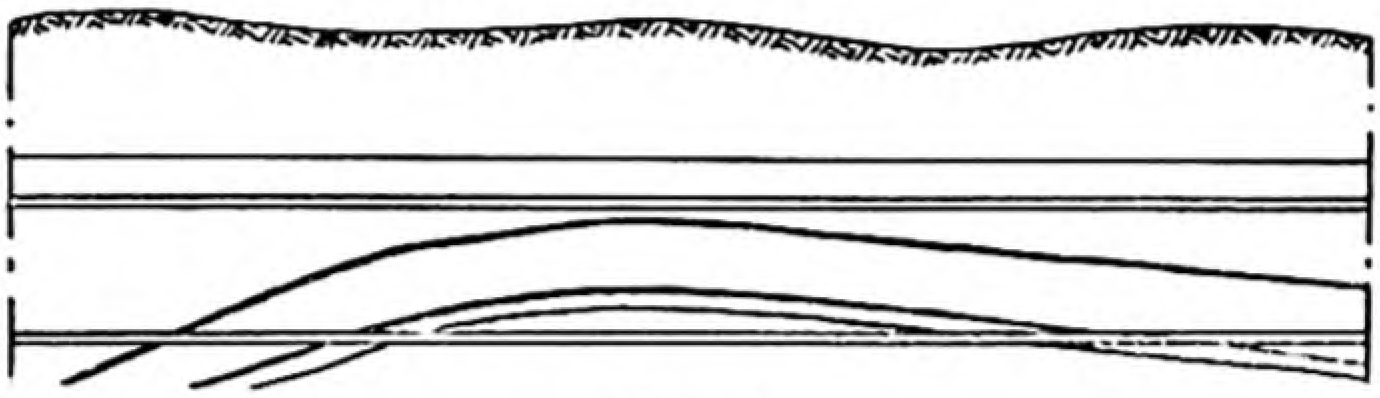


FIG. 6

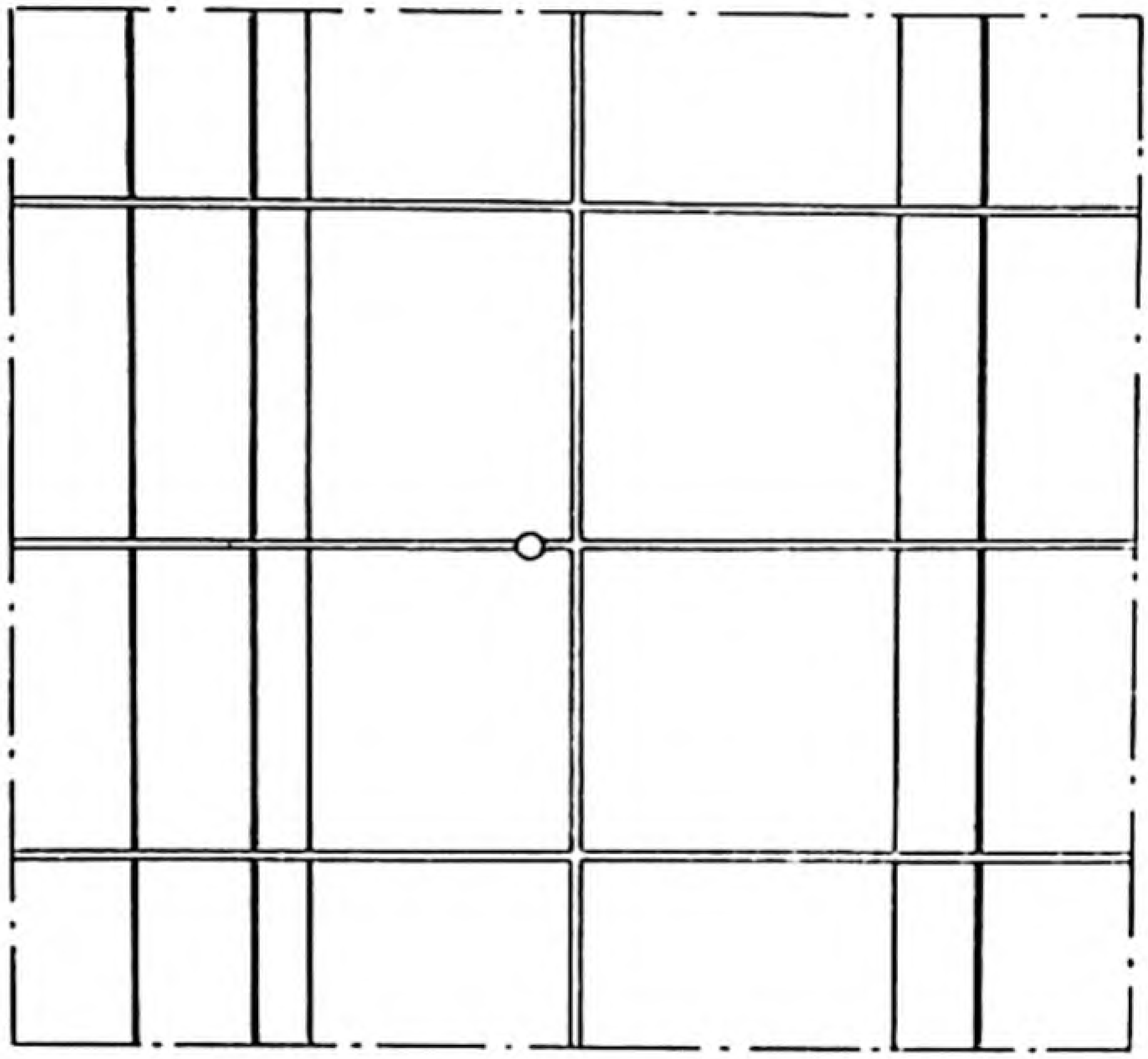


FIG. 7

the reserves and to provide suitable access to the seams at points other than those intersected by the main cross-cut. These further cross-cuts can be termed 'district cross-cuts', and also serve to distribute the ventilation by splitting the air into several parallel circuits. The same consideration applies to transport on these cross-cuts, which provide easier, quicker and cheaper transport on to the main

road haulage than would be the case if gate-road conveying was not limited.

Usually the cross-cuts are at right angles to the strike of the coal seams. Thus, if there is no change in the general direction of strike, these cross-cuts will be parallel. The reverse is the case in areas where a change in the general direction of strike occurs due to special or local synclines and anticlines. It should be endeavoured, if possible, to arrange that the seams are intersected by the cross-cuts more or less at right angles to the local strike of the seams, and in these cases it may happen that the cross-cuts are at an acute angle to each other.

In cases where local deviations in the direction of strike occur, the cross-cut may not cut the seam at right angles to the local strike, as in Fig. 8. The term 'cross-cut' is used if a road is at right angles to the general direction of the strike, irrespective of such local deviations.

The direction of cross-cuts will also be influenced by faults running at an acute angle to the general strike, in which case it may be considered advisable to equalise the reserves on each side of the cross-cut by setting out the cross-cut in an appropriate direction between the faults.

(c) *The distance between cross-measure drifts.* The distance between cross-cuts is an important problem which must be considered carefully. The greater the distance, the smaller will be the cost of drivage in rock and maintenance for that horizon network, but the points available for coal-extraction development are reduced, while the length of gate roads will be increased. With increased length of gate roads, the cost of auxiliary haulage will be increased, as will be the maintenance cost for these roads. The ventilation of the working faces will become more difficult due to the greater restriction in these roadways of smaller cross-section. The temperature rise will be greater, since the air is given a greater opportunity to come into contact with rock faces which have been recently exposed. On the other hand, where a greater number of cross-cuts are used, the length of the gate roads required will be decreased and the life of gate roads will be less, the length of expensive gate-road conveying will be decreased. The cost of drivage and maintenance of the stone roads will be increased. An optimum distance based upon local considerations should be derived. A method of appreciation of this problem is detailed later in Section 4 (*b*), in which the variables to be taken into consideration are described and certain suggestions are made.

Two of the main factors to be taken into consideration which affect the distance between cross-cuts are maintenance cost of gate roads and cost of gate-road conveying.

The maintenance cost of the gate roads depends upon the strata pressure and therefore on the type or form of support to be used, and on the anticipated life of the gate road. The methods adopted for the support of gate roads have been greatly improved over the last twenty or thirty years by the introduction of steel supports. These supports are normally either yielding or the hinged-girder type, with gate-road packs carried along each side. The rate of advance of the gate road has been increased so that the life of a gate road of equivalent length to that driven in earlier times is considerably reduced; this factor naturally tended to increase the distance between cross-cuts.

Gate-road transportation costs have been reduced by the introduction of conveyors and small locomotives in conditions where these can be used. The latter are used when gate roads are not straight and the output from the faces is not high enough to require a continuous haulage system, which conditions arise where the measures are steep and folded. This development has also tended to increase the distance between the cross-cuts, thus, the usual distance preferred in present-day Continental practice in flat measures is between 800 and 1,200 yards, as compared with 400 to 800 yards twenty years ago. The distance between cross-cuts will be further influenced by the position and direction of major faults, by which it is implied that extensions of the gate roads would not intersect the seams on the other side of the fault. As a rule, every block of strata limited by two diagonal faults requires a separate cross-cut, *vide* Fig. 8.

Where several faults exist within a working area, the distance between the cross-cuts may be reduced and the number of cross-cuts increased from what would normally be required in an undisturbed area of the same extent, *vide* Fig. 8. This is also particularly evident in the case illustrated in Fig. 9, which shows a working area of limited width in the direction of strike (about 1,300 yards), but having a long length across the strike. One main cross-cut from the shaft would have been sufficient, but for the presence of two faults running at right angles to the strike which reduce the working distance from the main cross-cut and require the addition of two further parallel cross-cuts to the west and east of the main cross-cut in order to develop the area.

The gradient of the seams will also influence the distance between cross-cuts. In many cases the rate of advance on faces in flat formations is higher than on highly inclined faces, principally due to the

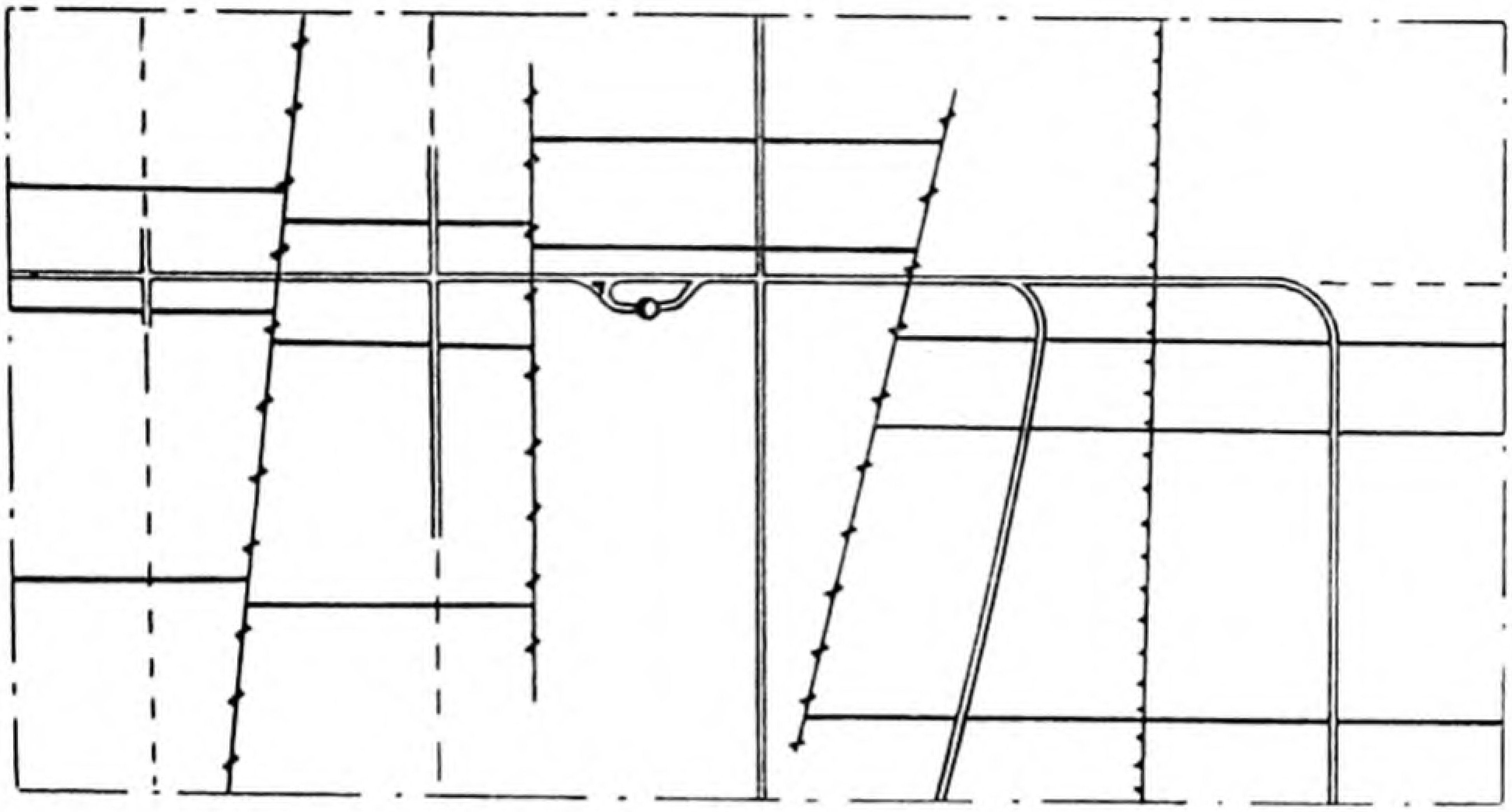


Fig. 8

method of stowing employed. In flat-seam working, strip-packing, caving or mechanised stowage methods can be applied, while stowage by hand is usually employed on steeply inclined faces. The number of

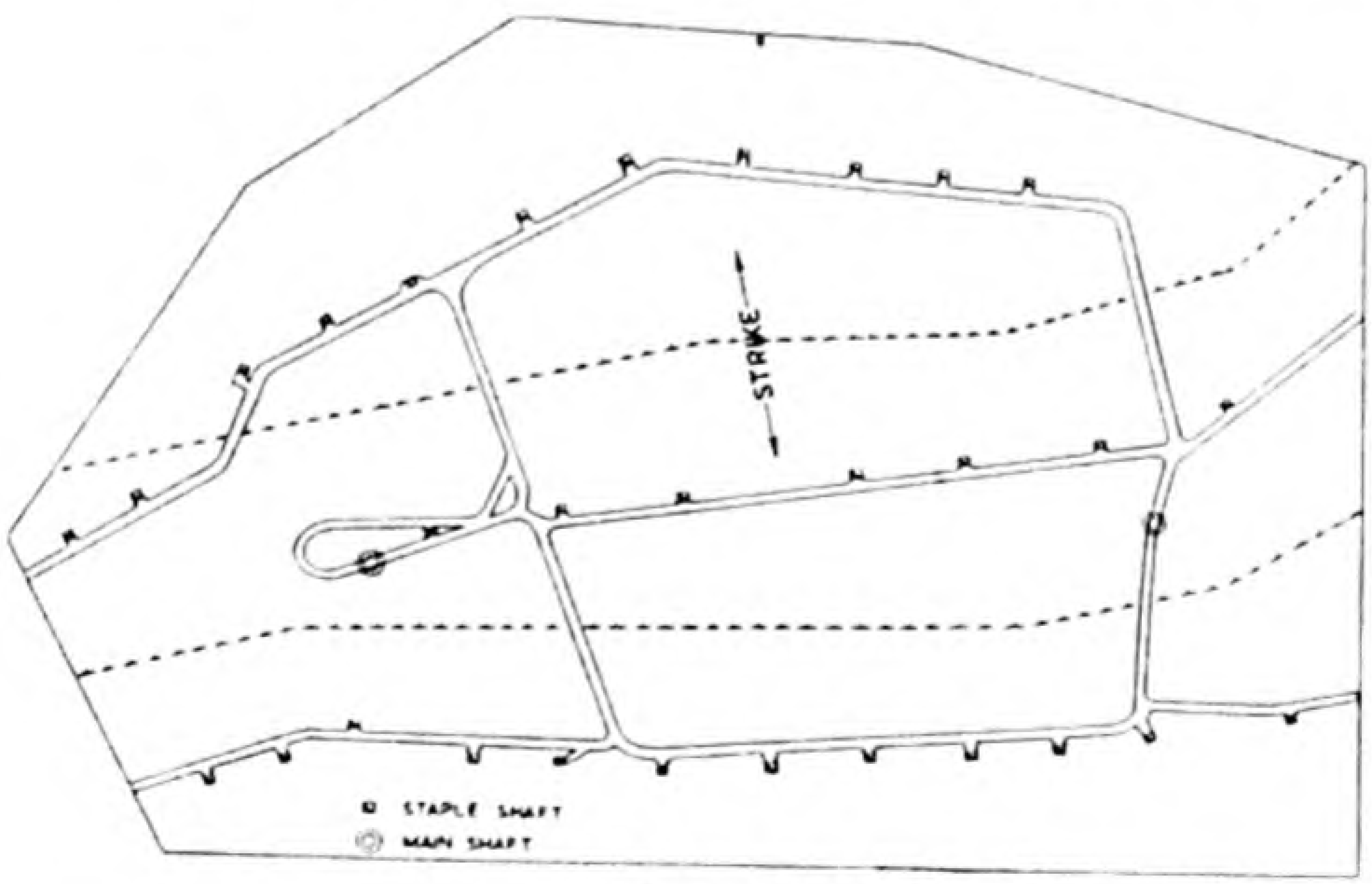


Fig. 9

men who can be employed safely on a steep face is less than the number who could be employed usefully on a similar length of face in a flat seam. The stowing capacity in a steep seam will depend upon the

tipping arrangements provided in the upper gate road, which may be 150–200 tons per shift or as high as 300 tons per shift. The stowage capacity of stowage machines is considerably greater than hand-stowing methods, also when caving or strip-packing is adopted a greater rate of advance can be achieved than is possible on a steep face with hand-packing. With a low rate of advance the life and maintenance costs of the gate roads will be increased, and therefore the distance between cross-cuts in steep measures are usually less than in measures of flat formation.

(*d*) *Converging and diverging faces and their influence on the distance between cross-measure drifts.* There are several methods of coal-face development from the cross-cut (not considering at present whether the faces are worked simultaneously or otherwise). If the seams strike west to east, one series of faces will advance east and the other west, 'diverging' from the cross-cut, whereas with north-to-south-striking seams the advance will be to the north and south respectively. The faces developed from two adjoining cross-cuts will advance towards each other, or 'converge', and will probably meet midway between the cross-cuts. The greatest length of gate road developed from any unit is therefore about half the distance between cross-cuts. The direction of the face line is governed by the cleat of the coal, especially where pneumatic picks are used. Where this occurs, it may be the case that one direction of advance gives better output results than from the other converging face, the difference in output per manshift being quite considerable. In such a case it may be advisable to have a greater length of development in one direction than the other, and the maximum length of the gate roads would be increased from the normal length of half the distance between cross-cuts.

Another system of development can be employed in which faces are developed from a single cross-cut and advance the full distance between cross-cuts, the faces between three cross-cuts being developed along the same panel and in the same direction. A further method can be employed in which a single face advances from the first cross-cut and past the second until it reaches the third, a distance of twice the length between them. If the development is planned as in the two latter methods, it is possible to choose the direction of advance most suitable to the prevailing conditions, and the team of men working on any individual panel is engaged

on the same unit for a longer period. The inevitable troubles experienced at the commencement of any unit have to be overcome only once within the working range between cross-cuts, with the probable benefit of an earlier realisation of a constant and maximum output per manshift. A disadvantage of those methods of development which incorporate a longer length of advance is the restriction of the number of working faces which may be developed between a series of cross-cuts, together with the longer length of gate road to be maintained and the consequent reduction of the distance between cross-cuts. If working towards both sides, twice as many faces can be developed than if working is confined to one side only. The cross-cut is probably not being used to its full coal transport capacity if the faces are confined to one side only; thus, since every cross-cut is not used to capacity with single-unit extraction, a greater number of cross-cuts are required simultaneously to produce the same daily output. This may be overcome by working faces in several seams to the same cross-cut, but the disadvantages of simultaneous extraction within the same area in adjacent seams make it sometimes difficult to consider this system as an adequate alternative. The sequential working of adjacent seams is discussed in detail at a later stage in Section 7.

A method to reduce the disadvantages incurred with single direction advance is to drive a pair of gate roads towards the advancing face and in the same line from the next cross-cut, so that the gate roads and face meet midway between the cross-cuts. The face continues in the same direction but is now retreating with the lower gate road as the mothergate, the upper gate road acting as a return. The same procedure can be carried out by driving a single gate road in the line of the existing mothergate from the opposite direction; this gate road takes over coal transport midway between the cross-cuts and eliminates the maintenance of the first part of the gate road. The ventilation is now taken through the mothergate and across the face to the upper, original return gate road, which is continued in the same direction. It is also possible to continue using the original mothergate intake gate road as an additional intake to the face where this is required due to high gas emission, the return gate road being continued as previously described as the face is retreated.

(e) *Main lateral roads.* The cross-cuts are connected to the shaft by means of main roads driven on the strike, these roads being driven

as straight as possible to facilitate haulage. The main roads may be in the seam or in the strata. Drivage in the seam reduces the cost and time of development, but usually increases the maintenance cost. Since these roads have a long life, the higher maintenance cost has to be borne for a considerable time and it is more economical to accept the higher drivage cost and keep the maintenance cost to a minimum. Continual maintenance work on these roads also interferes with their normal haulage duty. Thus, in general, lateral roads are driven in the strata, and in many cases are driven in the strongest measures even if the haulage distance to the shaft is increased. There are exceptions to this rule where the seams have particularly good roof and floor conditions, in which cases the laterals are driven in the seams. The number of lateral roads required for one level depends upon the extent of the area to be developed, the situation of the main winding shaft and the number of seams to be worked. If the longest length of development is to the strike, with the shaft located centrally in the area, one lateral direction is usually sufficient, with an extension both sides of the shaft on the line of strike as illustrated in Fig. 8. If the seams in steep, or semi-steep, formations occur in separate groups as shown in Fig. 10, two separate lateral drifts would be required, and especially if the largest extension of the area occurs at right angles to the direction of strike or if the main winding shaft is located near a boundary.

(*f*) *Lateral roads not on the strike.* As a rule, and according to definition, a lateral road will run parallel to the strike, but there are exceptions. Since the strike of the seams is not always regular, the lateral road may, in order to keep it straight, intersect the seams at an acute angle. In special cases, where the formations form a synclinal basin or local anticline, the lateral road may intersect the seams at right angles. The general strike of the seams in an area is considered as the reference direction, and if a main haulage road is driven in that direction, it would still be called a lateral, even if it intersects some of the seams in the cross-cut direction.

Section 3. Layout of the Return Airway Horizon

The principles upon which the layout of the return airway horizon is based are similar to those considered when laying out the main haulage horizon. The network consists of cross-measure drifts, lateral roads and some gate roads driven in the seams. Similarity in layout

can be easily understood when it is considered that the main haulage horizon ultimately becomes a return airway horizon in the second stage of its life, after a new and lower main haulage horizon has been developed. The return airway horizon is connected directly with the upcast shaft, and the connection with the downcast shaft is appropri-

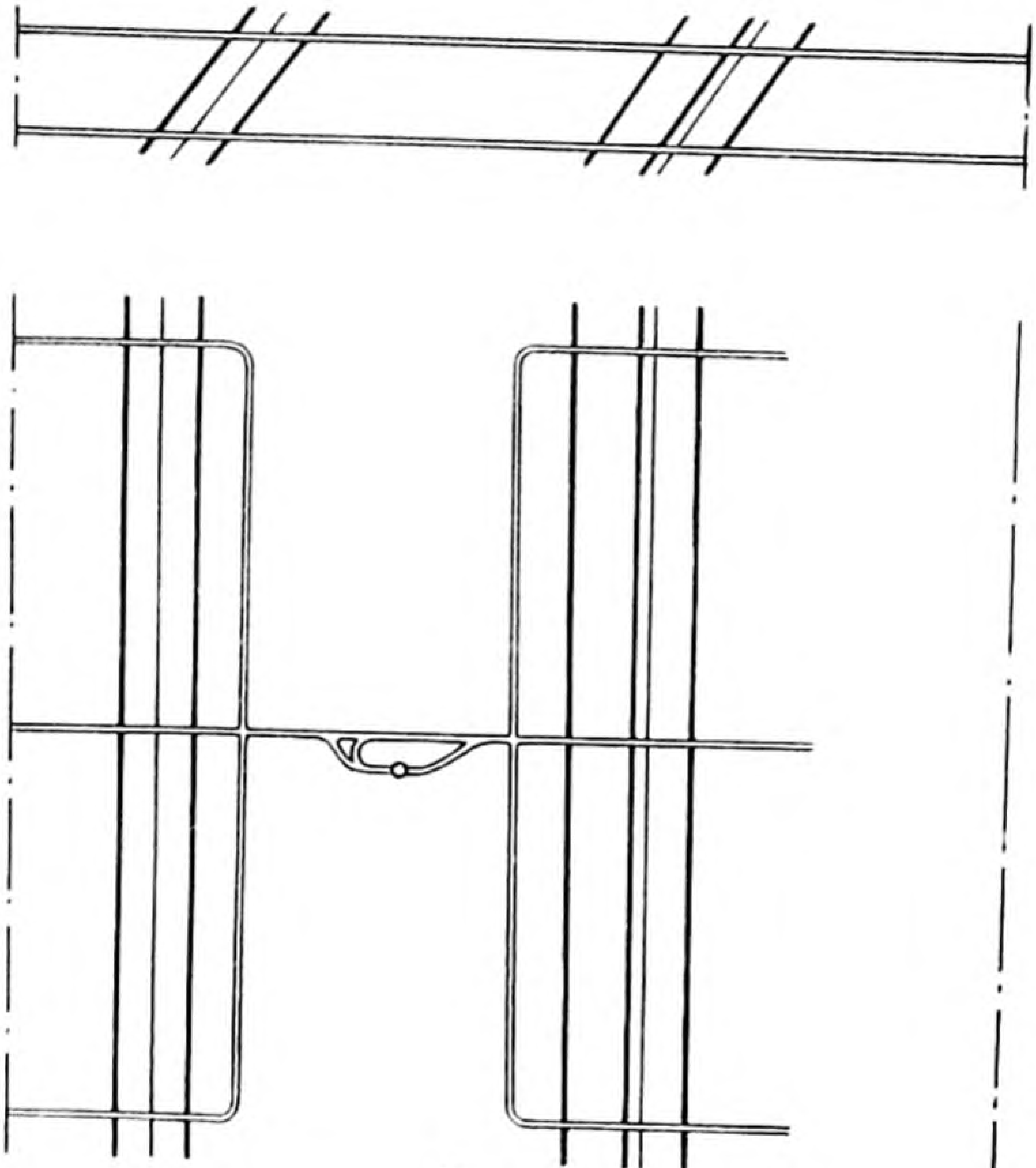


FIG. 10

ately sealed off from the return side, using the normal ventilation doors.

For the purpose of securing effective ventilation and to avoid additional costs in the return airway horizon, it is important to set out the cross-measure drifts in the different horizons immediately above one another.

It very often happens that, due to greater experience in technique, the distance between cross-measure drifts in the deeper horizons is greater than in the higher and older horizons. If it is possible to double this distance in the main haulage horizon in comparison with

that in the return airway horizon, then, every second cross-measure drift in the upper horizon corresponds in position to one in the lower main haulage horizon, and the condition stated previously is fulfilled. The intermediate cross-measure drifts are no longer required and become redundant. Where the distance between cross-measure drifts in the new haulage horizon is less than twice that in the airway horizon, only a few of the upper cross-measure drifts correspond in position with those developed in the lower main haulage horizon. In this case it is necessary to drive new lateral gate roads, or maintain existing gate roads, which will serve as connections between

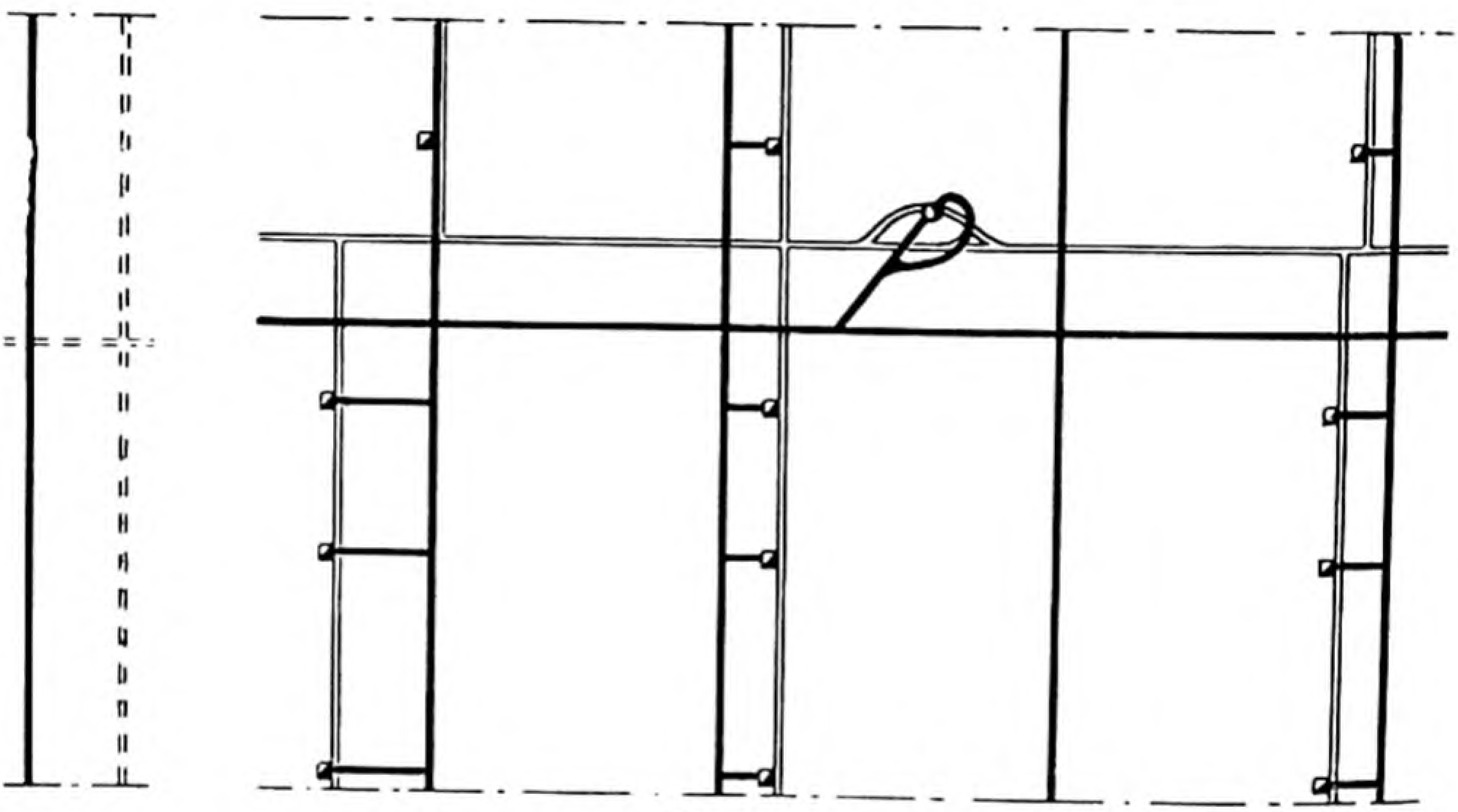


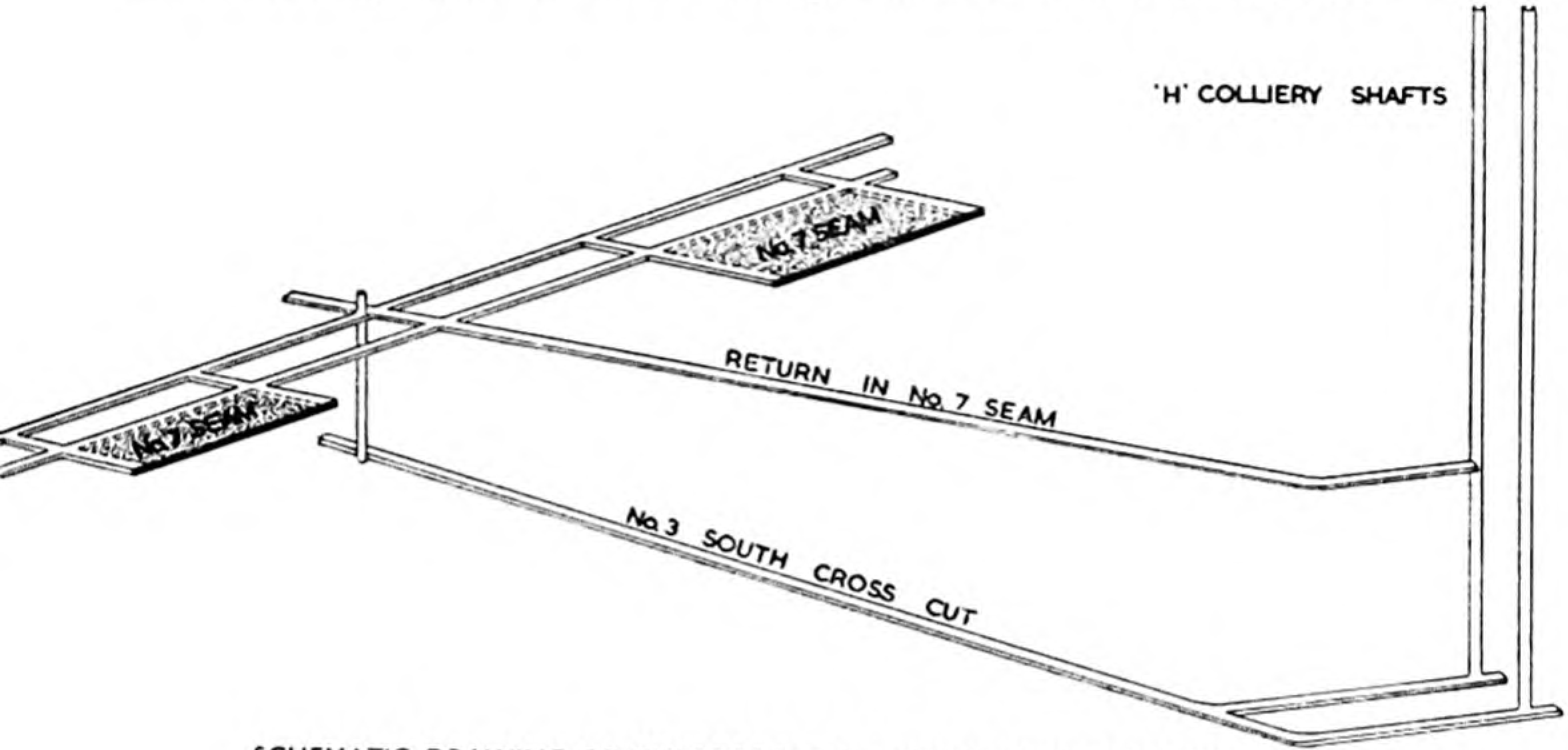
FIG. 11

the vertical staple shafts between the two horizons and the next cross-measure drift of the return airway horizon, *vide* Fig. 11. (Refer to Part II, Section 4.)

In the general reorganisation of an existing colliery, where the system of horizon mining is to replace development by the conventional single-horizon system, generally it will be possible to utilise existing roadways in an upper seam or seams, as a network for the return airway horizon. Where it is possible to plan the development in the new haulage horizon to allow main roads to be driven immediately beneath existing roads above, this should be done, otherwise similar considerations must be made with regard to providing suitable connections between the main haulage horizon and

the return airway level. Fig. 12 illustrates a reorganisation scheme initiated along these lines at a British colliery.

In a new colliery the return airway horizon must be driven simultaneously with the main haulage horizon. If the contact be-



SCHEMATIC DRAWING OF UNDERGROUND LAYOUT AT 'H' COLLIERY

Fig. 12

tween the upper immediate strata and coal-measure formations is level, or if the overlaying strata is of equal thickness, there is no special difficulty in the layout of this horizon, except in as much as adequate cover is required below the surface. If, however, the line of

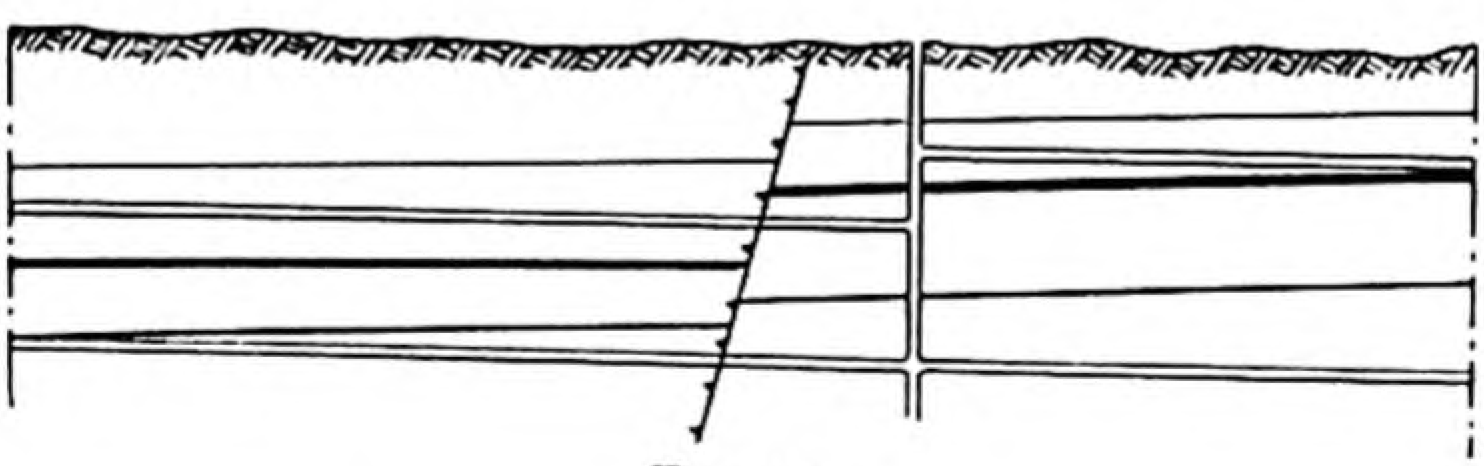


Fig. 13

contact dips, or if faulting occurs, it will be necessary to separate the return airway horizon into two sections, one of which is at a higher level than the other, as shown in Fig. 13. If this is not carried out, some of the upper seams, in whole or part, may lie above the return airway horizon, with consequent difficulty in ventilation due to the necessity to resort to descensional ventilation.

In general, in flat measures the first return airway horizon should be situated above and as near as practicable to the upper seam, whereas in steep measures this horizon should be as high as possible to eliminate any possibility of sections of the seams intersecting the return airway horizon.

Section 4. Layout between the Horizons

(*a*) *General remarks.* A lower main haulage horizon and an upper return airway horizon do not generally give sufficient points of attack for full development of the intervening seams. They do not normally divide the reserves into sections of a suitable area for extraction development. If the coal-measure strata is flat, it may be the case that some or all of the seams do not cut the horizons and are not intersected by the cross-measure or lateral drifts. The development between horizons must therefore be supplemented by vertical connections. This is carried out by sinking staple shafts between the horizons. These shafts are also called 'blind' shafts or 'staple pits', and serve for winding coal, men and materials, and for providing a ventilation connection with the return airway horizon.

(*b*) *Staple shafts in flat measures and in cases where faces are developed on the strike.* A cross-section through the strata (in which there are three almost level seams) between two cross-measure drifts at different horizons is shown in Fig. 14. The objective is to subdivide the series of seams into working blocks or sections, and to provide means for transportation between the seams and the main haulage horizon. This is done by introducing staple shafts at suitable points and at an adequate distance apart. The most suitable distance between staple shafts depends in the first instance on the most convenient length of section into which the reserves can be sub-divided, and this length is identical with the length of coal face won out in each of the several seams. The decision as to the best length of face to be used is an important one. If there are several seams each of different section and differing in the essential physical characteristics of the seam, roof and floor, a compromise must be made between the most favourable length of face in each of the individual seams. This length is not generally difficult to determine, since it will depend more on the capacity and type of face machinery to be installed, and on the system of support applied, especially on the method of packing adopted, than on the physical nature of the seams to be worked.

In Fig. 14 the total length of the cross-measure drifts is 1,200 yards. If 200 yards is considered as a convenient length of coal face, seven staple shafts would be required. Two staple shafts would be situated at the boundaries of the section, and the remaining five spaced at 200-yard intervals.

The distance between staple shafts will also be influenced by the

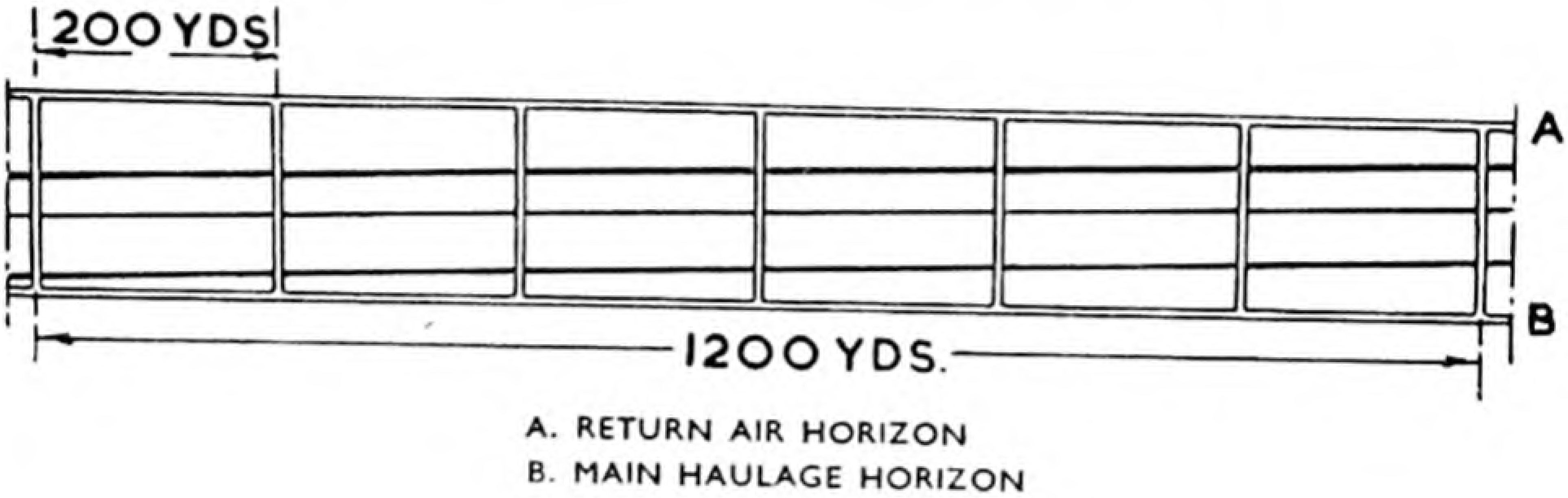


FIG. 14

presence of faults running in the direction of the strike, these influencing the length of face which can be developed between the fault positions, and full development will be changed to suit these conditions. While it may still be possible to provide the same 200-yard

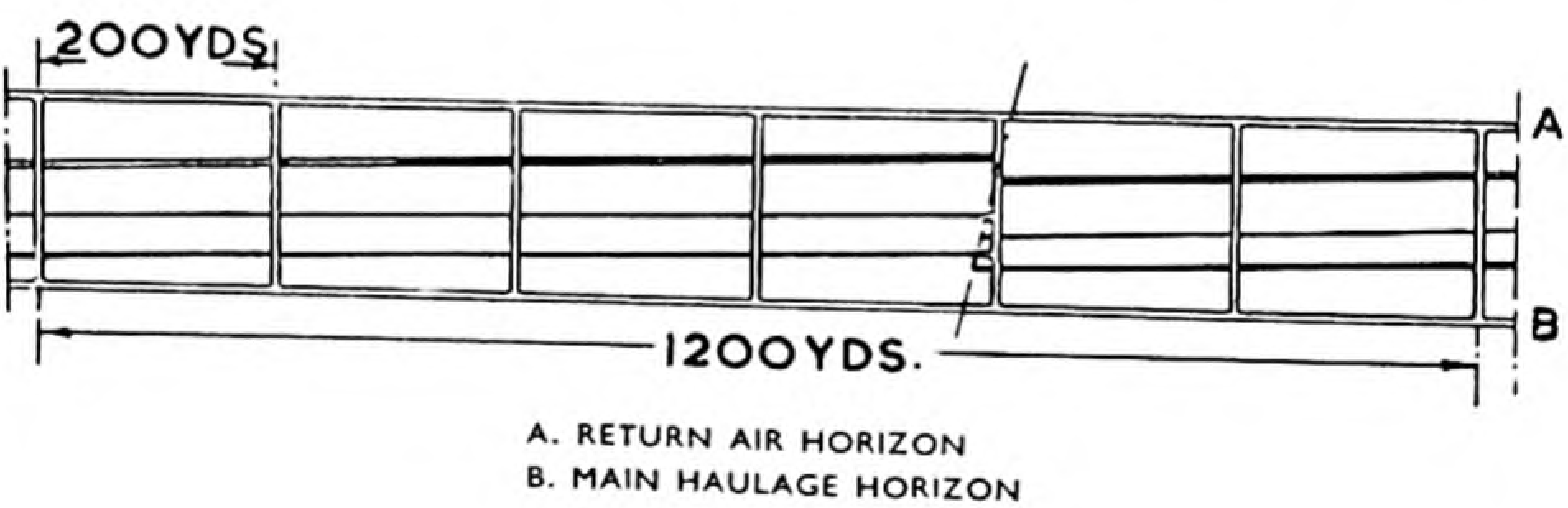


FIG. 15

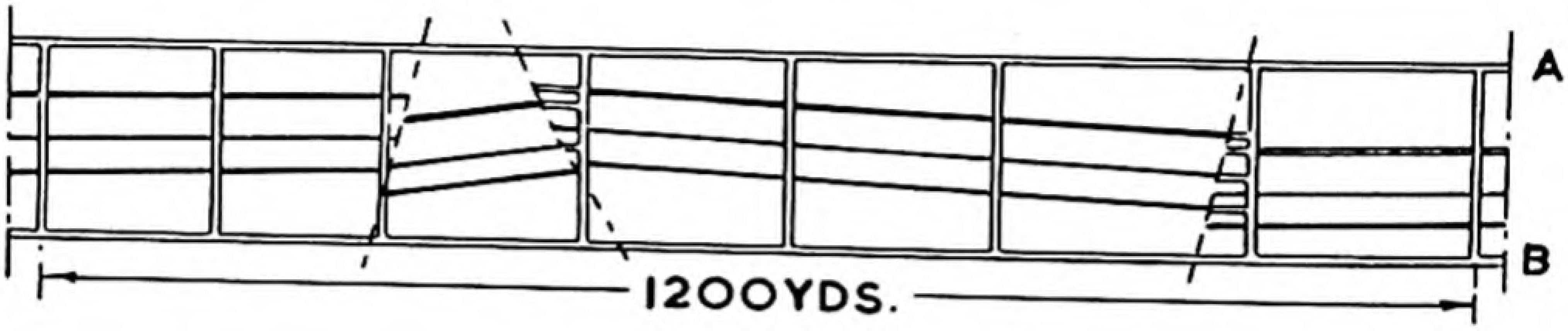
spacing in a faulted area, as illustrated in Fig. 15, it is probable that the fault locations do not suit a standard face length; consequently, the number of staple shafts required will be greater but their distance apart will be less. This is illustrated in Fig. 16.

Another example of the influence of faulting on the general layout of the staple shafts is shown in Fig. 17. The cross-section shows semi-steep and flat seams dislocated by faults which run parallel to

the line of strike. The faults sub-divide the district into five different zones in which there are a different number of seams between any pair of horizons.

The layout of the staple shafts between the two lower horizons can be discussed in detail.

In zone 1 the measures are flat so that longer faces can be laid out



A. RETURN AIR HORIZON
B. MAIN HAULAGE HORIZON

FIG. 16

and developed than in the semi-steep zones, and for this reason it is possible to have a greater distance between staple shafts. A short staple shaft from the lower horizon to seam 6 is required in order to provide access for intake air. The position of the first staple shaft is governed by the extension of working in seam 6, and the upper

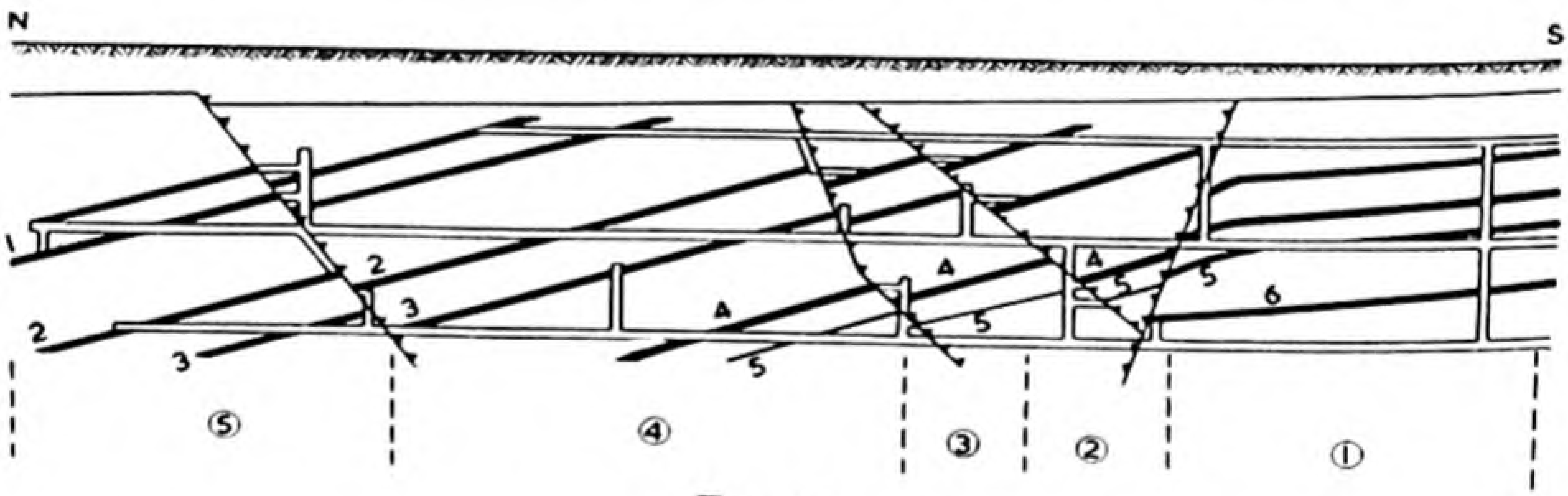


FIG. 17

seams to the south end of the section. Seams 4 and 5 in zone 2 can be worked now. Seams 4 and 5 do not offer any special difficulty in working, the loading-point being situated in the lower horizon at the bottom of the third staple shaft, that section of seam 5 on the upthrow side of the fault in zone 1 can be worked with seam 5 in zone 2.

Seams 4 and 5 in zones 3 and 4 may be considered together. Seam 4 in zone 3 is worked first. The lower end of seams 4 and 5 are con-

nected to the intake level by means of a short staple shaft; similarly, the upper ends of the same two seams are connected to the return airway level by the third staple shaft, which has been located between the two main horizons when considering zone 2.

Seam 5 in zone 3 is worked next in the sequence, followed by seams 4 and 5 in zone 4. During this latter period a rise maintained in seam 4, zone 3, serves as a return airway in conjunction with the upper part of the second staple shaft driven between the middle and lower horizons. In zone 4, the intersection of seam 3 by the middle and lower horizons is too long to develop as one face, therefore an intermediate staple shaft is driven to intersect seam 3 from the lower horizon. This fourth staple shaft serves for coal transportation to the lower level, and as intake airway to the high side coal, which is worked first. The low side coal is worked from the intersection with the lower main haulage level and the loading-point is transferred to the point where seam 3 intersects the lower horizon. The return air from the low side working is carried to the return airway level by a rise maintained on the high side. Seam 3, zone 5, is considered with the development of seam 2 in zone 4, which is extracted prior to working seam 3. The lower end of seam 2 must be connected with the intake air level, and this is done by means of the staple shaft and short sub-level cross-measure drift. A pair of faces is developed in seam 2 by driving a rise in the seam from this point to intersect the middle cross-measure drift. This rise is maintained after seam 2 is extracted, while seam 3 in zone 5 is being worked. This rise serves as the return airway for the faces in seam 3 in conjunction with the staple shaft and sub-level drift. It is important to note at this juncture that the loading-points for seams 2 and 3 in zone 5 would be located in the lower level.

In zone 5 there are three seams to work. Seam 1, which is isolated from the two lower seams, does not justify the driving of the lower main cross-measure drift beyond the intersection with seam 2 or the sinking of a staple shaft upwards from the lower level to this seam only. For this reason a short staple shaft is sunk from the middle cross-measure drift, and ventilation is brought to the face through this staple shaft. The extraction of seam 1 is carried out during the life of the middle horizon as a main haulage and intake air level. In seam 2 the probable length of the face is 150 yards between the lower horizon and the fault. It is necessary to connect the upper end of the seam with the return airway horizon. This can be done by a staple

shaft or by an inclined drift in the rock on the line of the hade of the fault, and in Fig. 17 the latter method of development is illustrated.

(*c*) *Staple shafts and sub-levels in steep measures.* The staple shafts fulfil the same functions in steep measures as has been described for semi-steep and flat measures, but there are some notable differences. They serve a greater number of seams in most cases, due to the folding of the formations, and they do not directly sub-divide the reserves between the main horizons in working sections of adequate size. This task is fulfilled by the development of sub-levels.

Sub-levels may be defined as auxiliary horizons between the main horizons. They are, in fact, mainly cross-measure drifts which are driven from the staple shafts in order to cut the various seams and to give access to them. The sub-level cross-measure drifts are not con-

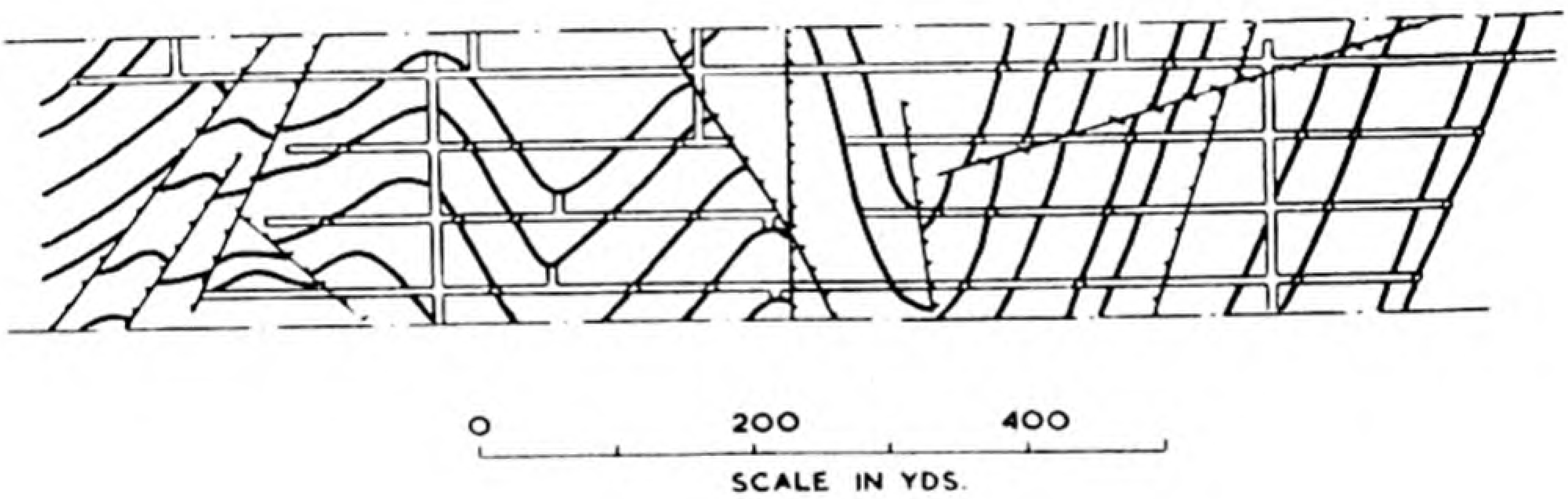


FIG. 18

nected to each other by lateral stone drifts or directly to the main shafts, but are entirely independent levels within each cross-measure drift district. The manner in which the sub-levels divide the reserves between the main horizons is illustrated in Fig. 18. In this section there are nine steeply inclined seams. The distance between main horizons is 180 yards, and this distance is too long for the development of individual faces of such a length between horizons. These faces run to the dip, and the actual face length is longer than the vertical distance between horizons, being greater when the faces are laid out diagonally to the full dip. Vertical heights of from 50 to 100 yards are considered reasonable in steeply inclined seams, which mean a sub-division into two to four sections. In the case illustrated in Fig. 18, 60 yards has been considered reasonable, mainly due to the existence of small faults not shown on the section, and so making two sub-levels necessary.

The number of staple shafts required in steep measures is much less than in flat measures, and in the case of the formation shown in Fig. 18, only two staple shafts are necessary at a distance of about 580 yards apart, serving a total length of cross-measure drift of 850 yards. The reduction in the number required is due to the fact that the staple shafts are connected by means of the sub-levels to a greater number of seams. If this system were applied to flat measures, unnecessarily long auxiliary cross-measure drifts (sub-levels) would be needed, and these would only cut a small number of seams, so that the capital expenditure and working costs would be prohibitive.

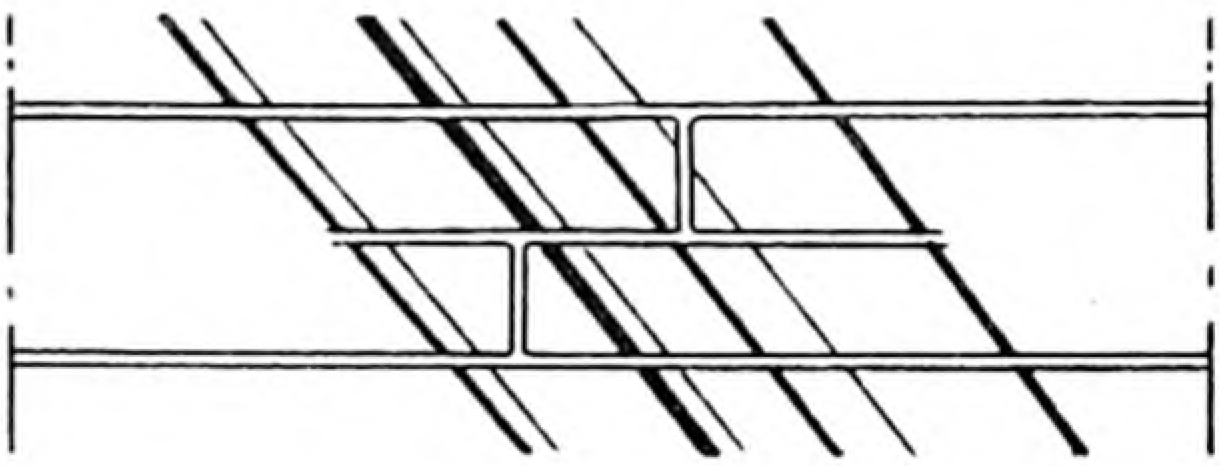


FIG. 19

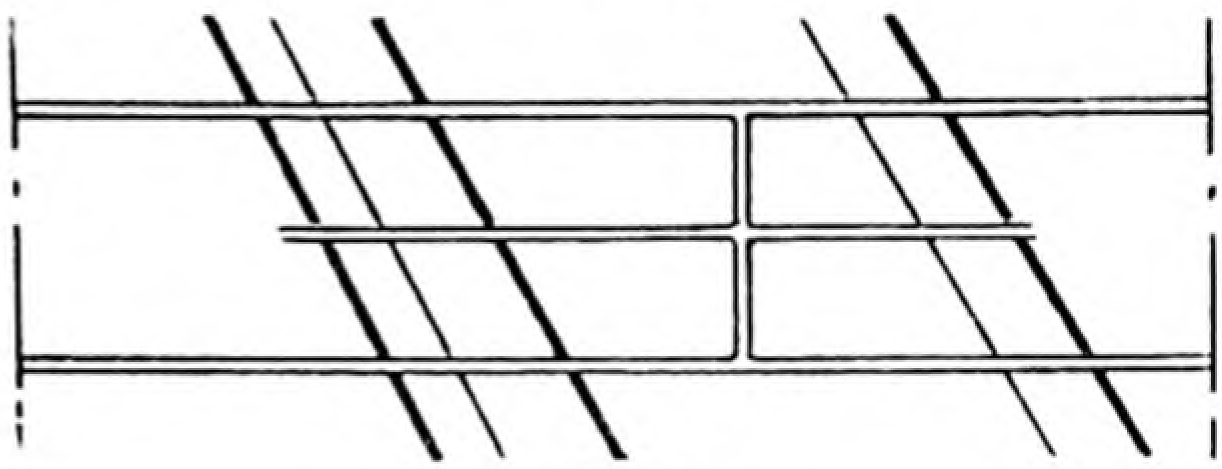


FIG. 20

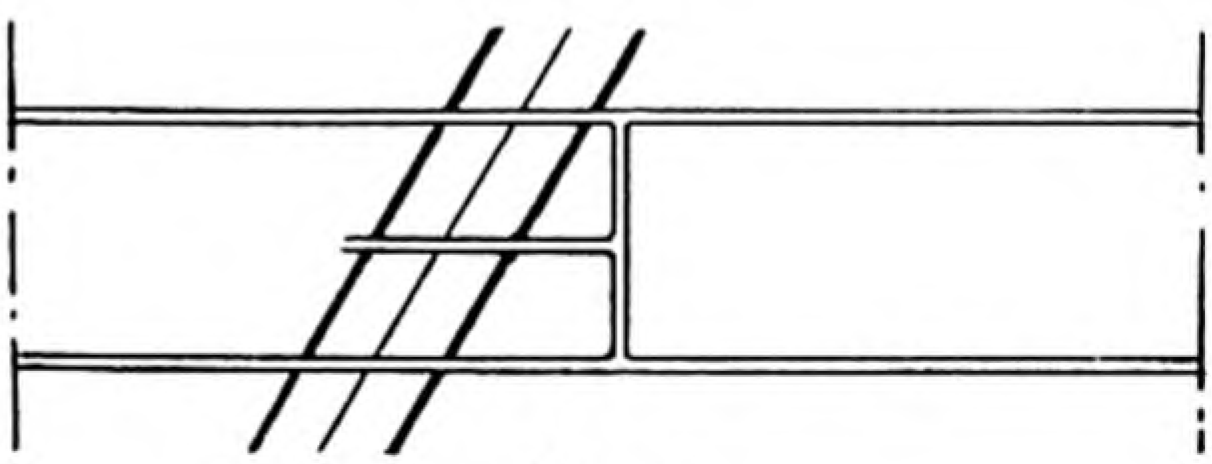


FIG. 21

(*d*) *Position of staple shafts in relation to the seams in steep measures.* An important problem is to decide whether the staple shafts should be located within a group of seams, as shown in Fig. 19, or outside the seam formation and in the intervening strata. The advantage of the former position is mainly due to the short sub-level drifts required and the consequent low sub-level haulage costs. The main disadvantage lies in the fact that the shaft is affected by the coal extraction and has to withstand severe strata pressure. It is possible to overcome such difficulties, as explained in Chapter 2, and to maintain the staple shaft, even if the whole of the coal reserve is extracted within its immediate vicinity. These difficulties are definitely less if the shaft is located in the strata between the seam formations, as in Fig. 20, or outside the formation, as in Fig. 21. In the latter case the

location has the disadvantage of high haulage costs due to the length of the sub-level.

In general, if the geological conditions allow it, the position of the staple shaft should be chosen outside and on the floor side of the seams. The stability of the staple shaft is more important than the slight gains arising from a lower haulage and drivage cost on the sub-level. In many cases, however, the location of the staple shaft within the group of seams cannot be avoided (Fig. 19).

It is also better to have a single staple shaft connecting both lower and upper horizons, than to sub-divide the shaft into two sections. In this case, the upper section connects the sub-level to the return airway horizon and the lower section to the main haulage horizon, as shown in Fig. 19. The advantage of such a layout lies in the simple arrangement for two-cage winding which can be applied in each staple shaft, but this requires two separate winding units and doubles the work involved in the handling of stowage material and other supplies for coal faces between the sub-level and return airway level. If no other considerations have to be made, one single shaft is preferred.

(*e*) *The working of isolated sections between horizons.* Due to faulting and other geological conditions, isolated strips of reserves may occur in positions between horizons or below the main haulage horizon, and these have to be given special consideration. These sections of seams are developed by small staple shafts either from the upper or lower levels. Reference should be made to Fig. 17, in which such a development is applied to seam 1, zone 5, and to Fig. 22, where the development of isolated synclinal sections is illustrated.

(*f*) *Long staple shafts or short staple shafts.* The general considerations upon which a decision on this point should be made have been discussed already. It should be noted, however, that where geological conditions permit, many mining engineers prefer to have staple shafts which connect directly from lower to upper main horizons. This arrangement is preferred because of the ease of access to and from the faces, which can be reached from either above or below the seam level.

(*g*) *Inclined stone drifts replacing staple shafts.* Short staple shafts may be replaced by inclined stone drifts. The advantage of this system originally consisted in not having to use tubs in the gate roads and in having the main loading-point for coal or the tippler

for stowage material in the main horizon. In this case the coal or stowage material is transported by the normal belt or scraper or steel-plate conveyor, depending upon the gradient of the drift. Since

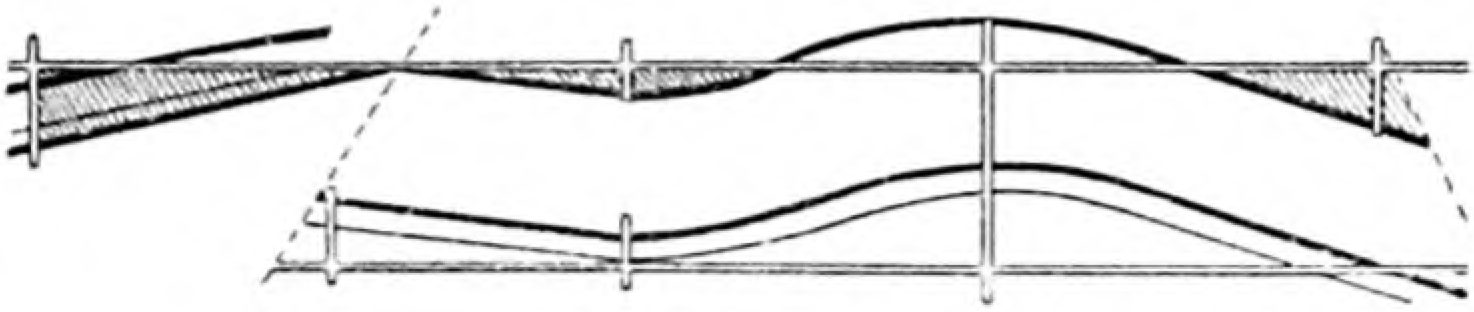
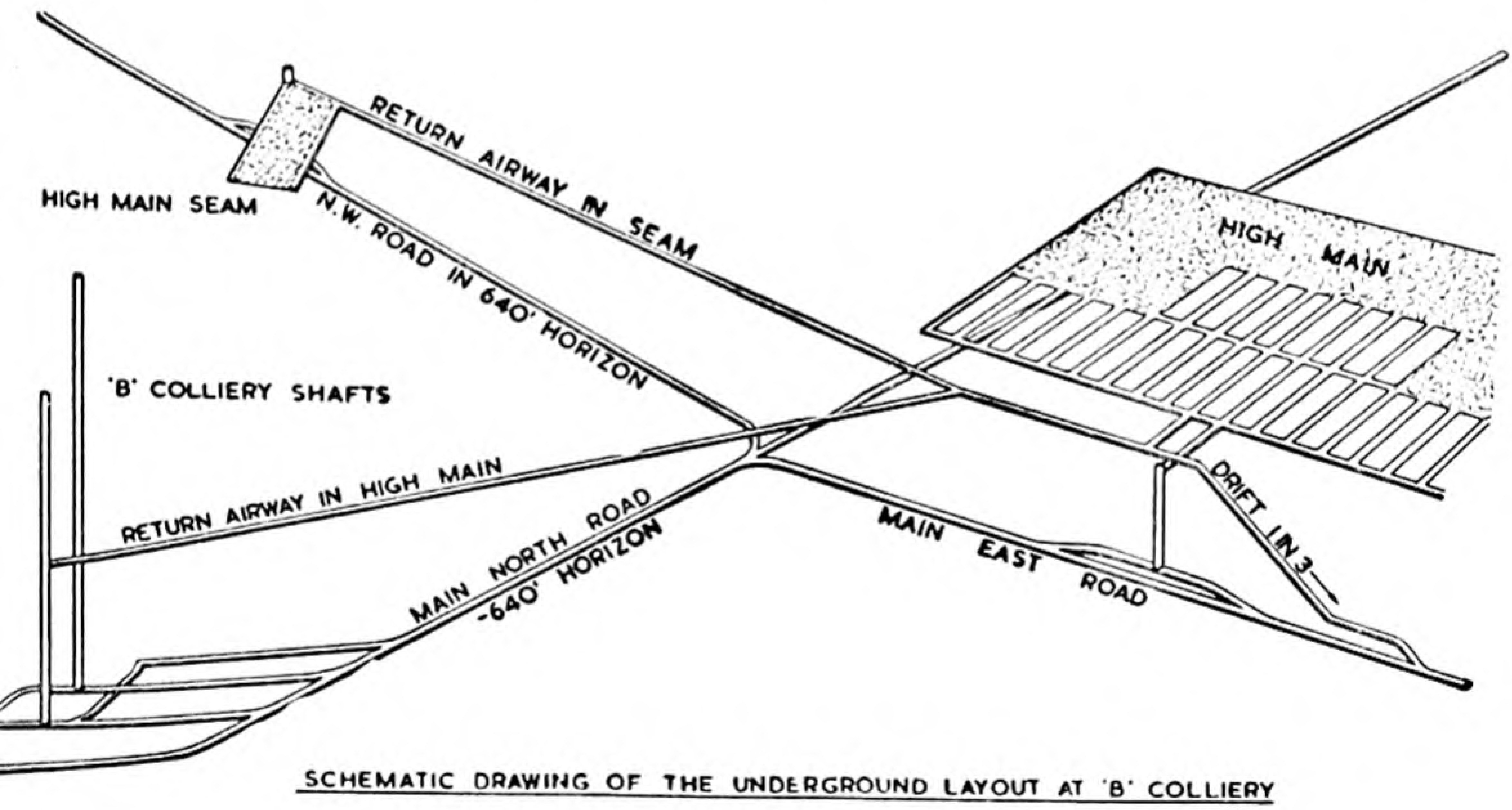


Fig. 22

the advent of the spiral chute for vertical conveying from the upper to the lower horizon, these advantages have been reduced and the choice lies in the individual preference for cage winding of supplies and stowage material in the staple shaft or by inclined drift. In any



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'B' COLLIERY

Fig. 23

case, the length of the inclined drift is correspondingly greater than the staple shaft; thus, the inclined drift is only an advantage if the vertical distance to be overcome is comparatively small, say, in the region of 50 yards. Reference should be made to Fig. 23, which

illustrates the adoption of an inclined supplies drift in a development at a British colliery.

Section 5. Distance between Main Horizons

(*a*) *General remarks.* The correct distance between the main haulage and return airway horizons is extremely important, and in order to make a decision several factors have to be considered. It is obviously a major decision of greater fundamental importance than the fixing of the distance between staple shafts and district cross-measure drifts, which distances may be varied to suit individual or local conditions and can be revised at any time according to the progress of mining technique. The location of staple shafts and cross-measure drifts may be planned and introduced during the course of normal mining development, whereas the main horizons, once they are planned and have been driven, remain fixed for the duration of mining activities over a long period. Only two other decisions are of greater importance; the location of the main shafts and their diameter. The capital outlay on a horizon, including the necessary staple shafts, must be redeemed by the extent of the reserves and the tonnage which will be obtained from this horizon and transported through its roads to the main winding shaft. It is obvious that there is a distance between horizons at which mining costs will be a minimum. These costs will include:

(i) Development of the network of roadways and staple shafts belonging to the new horizon.

(ii) The maintenance of these roadways and shafts.

(iii) Winding of coal and other materials in the staple shafts.

(iv) Winding in the main shaft.

(v) Ventilation.

(vi) Pumping.

(vii) The time for men riding.

The development cost of the cross-measure drifts and laterals of the new horizon are almost independent of the distance between horizons, since it is assumed that the nature of the rock in depth is the same. Temperature increase with depth may have a certain influence, especially in drifts ventilated by auxiliary ventilation when mining at greater depths. The cost of sinking staple shafts and other costs will both be higher, the greater the distance between horizons. The maintenance cost of roadways driven in rock or coal is influenced by

the distance, since strata pressure increases with depth. In addition, due to the increased reserves between horizons, the life of the roadways is longer, and consequently a higher maintenance cost has to be borne over a longer period, even though the total tonnage available between horizons is higher. Since the maintenance of a roadway very often increases over the period of its life, there are many instances in which it will be preferable to limit the life of a horizon to from fifteen to twenty-five years, as is the normal practice in the Ruhr.

The increased winding cost with deeper staple shafts is understandable. Where cage or skip winding is adopted in preferance to spiral chutes, the capacity is reduced. With spiral chutes, the capacity is not affected with increased depth of shaft, but the degradation which occurs is dependent upon the depth and the amount of fines produced is consequently increased. Mainshaft winding costs increase with depth, and in the Ruhr coal field it has been found that the increase amounts to approximately 2*d.* per ton per 100 yards between 450 and 770 yards, and 4*d.* per ton per 100 yards between 770 and 1,100 yards. The ventilation cost must be sub-divided between the maintenance of airways and the running cost of the main fan. The maintenance cost of the airways is influenced by the distance between horizons because of the reasons given previously, and since it may be more difficult to maintain these airways over a very long period. The main fan costs are influenced by the depth of the main shafts, since the pressure losses in the main and staple shafts increase with greater depth. The increase of temperature with depth may have an adverse effect on the working conditions underground, requiring an increase in ventilation to improve these conditions, with a consequent increase in the ventilation cost. Pumping costs must also be considered when deciding upon the distance between horizons; with increased distance this factor becomes more important, especially if the volume of water to be dealt with is large.

(*b*) *The influence of the coal reserves.* Since all these development and maintenance costs must be charged to the output obtained, the cost per ton of output is affected. If the coal reserves to be worked from one horizon are large, a high daily output is possible. If these costs are distributed over a high tonnage the charge per ton is small, and the charge is increased with decreasing coal reserves and decreasing annual output. In extreme cases the charge may become so

high that either another system of development must be adopted or the coal cannot be worked economically. For these reasons, it is obvious that the extent and quantity of the reserves at a colliery must have a great influence on the distance between horizons and, indeed, exert the greatest influence of all on the final decision. Should the coal reserves be high, the most favourable distance will be less than if the reserves are small. In the former case, there are sufficient reserves contained in a short vertical interval to redeem the development charges, whereas in the latter case the same quantity of coal is only available in a greater vertical distance. In other words, 'seam density', or the proportion of total thickness of coal to strata, plays a major part in deciding on the most appropriate distance to be adopted.

There is, however, no direct relationship between either 'seam density' or reserves and the most favourable distance between horizons. In other words, if the quantity of coal present within 100-yards range in depth at colliery 'A' is the same as within 200 yards of strata in colliery 'B', and the most favourable distance in colliery 'A' was found to be 120 yards, it does not necessarily follow that the appropriate distance at colliery 'B' would be 240 yards, but would be less. The costs which have been discussed already would require to be considered, and it is probable that a distance of about 175 yards would be chosen and the relatively higher development costs at colliery 'B' could not be avoided.

(c) *The influence of the seam distribution.* The seam distribution within the coal-measures also has an influence on the distance between horizons. If the seam sequence is repeated regularly in depth, this factor has little importance. If, however, the seams occur in groups irregularly in depth, as in Fig. 24, it is advisable to locate the main haulage horizon at such a depth that it is possible to work the lowest seam of any group from that horizon. Should this not be done, increased winding and other costs are incurred due to the greater distance between the upper group of seams and the next horizon; this condition is shown in Fig. 25. A method of overcoming this difficulty is by 'under-level working' which is described in Section 11.

(d) *The influence of the character of the strata.* A generally accepted rule is that it is preferable to have low working costs even if the initial capital outlay is high. In applying this rule to the choice of

horizon interval, it is better to have the shaft bottom and the greatest proportion of the main roadways in good solid strata, sandstones being preferable to shales. Whereas drivage of the shaft bottom in sandstones is more expensive than in shales, the maintenance costs are appreciably less, and drivages in such strata should be chosen if possible. If the main laterals and cross-measure drifts can be driven

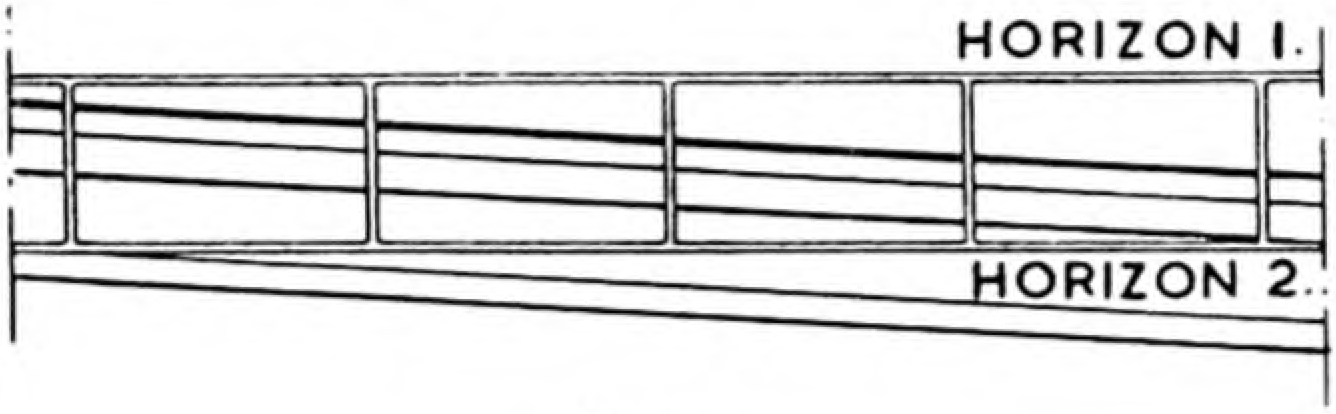
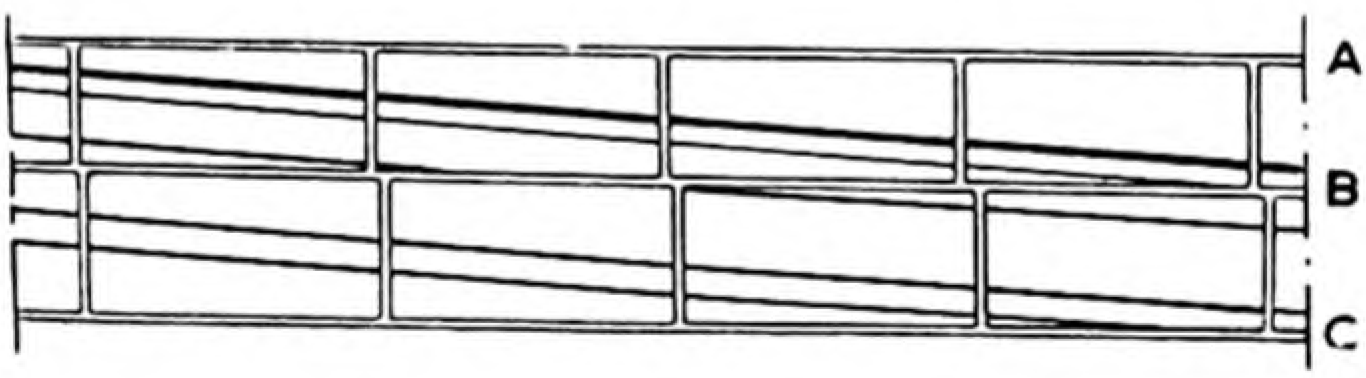


FIG. 24

in the same strata, this should be done. Variation in the dip of the beds may alter the general layout and, with increasing inclination, a decreasing proportion of the main roadways can be driven in the same bed. If the measures are very steep, it is probable that only part of the shaft bottom can be laid out in strong beds. This factor of



A HORIZON 1.
B " 2.
C " 3.

FIG. 25

rock character, however, is only of secondary importance and may influence the depth of the horizon to only a small extent.

(e) *The influence of the length of face and method of working.* It has been previously explained that in flat measures the length of face has an influence on the distance between staple shafts. This factor has no bearing on the distance between horizons where the same conditions exist, but in semi-steep or steep measures the length of face which can be worked has a considerable effect.

With increasing dip, it is possible to have one face extending from the return airway to the main haulage horizon. If, for example, the seams have an inclination of 35 degrees, coal faces of 150 and 200 yards in length correspond to vertical intervals of 85 and 115 yards. The distance between horizons could be either the same or twice this length. A distance of 85 or 115 yards is a comparatively short dis-

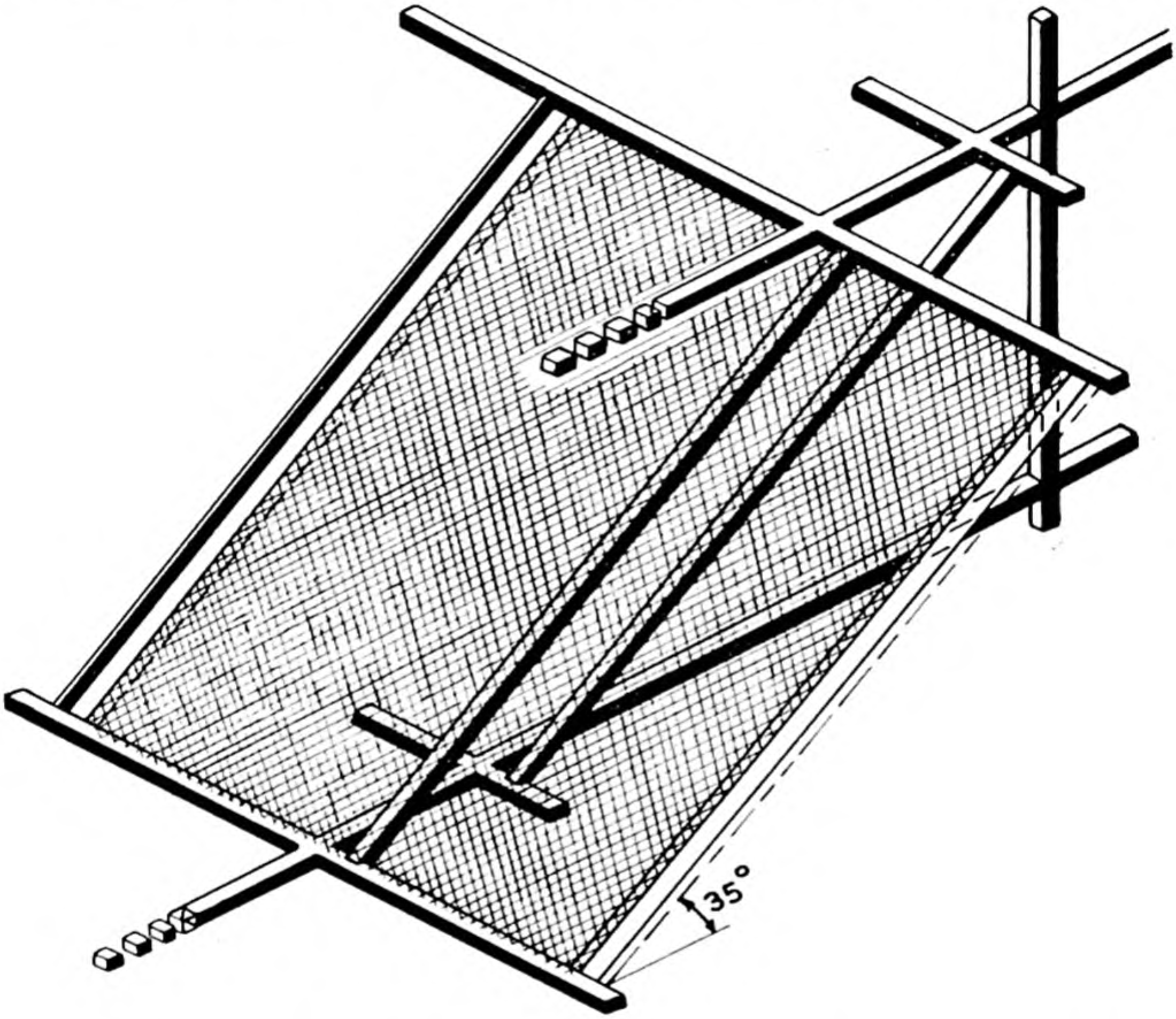


FIG. 26

tance between horizons, but it has the great advantage of avoiding sub-levels as in Fig. 26. In this case the development work in rock requires to be done on main horizon roadways only and sub-level drivages in rock are eliminated. The transport of stowage material along the return airway horizon must be arranged, if the requisite haulage arrangements are not available from the main shaft along this horizon. At least one staple shaft is required in every district to bring the stowage material from the district cross-measure drift of

the main haulage horizon to the corresponding district cross-measure drift of the return airway horizon, from which it is distributed to the adjacent working faces. It is possible, under the same geological conditions, to arrange two independent faces in line with each other and having a common mothergate, in which case the horizon interval is twice the vertical height of each face. If the individual face height

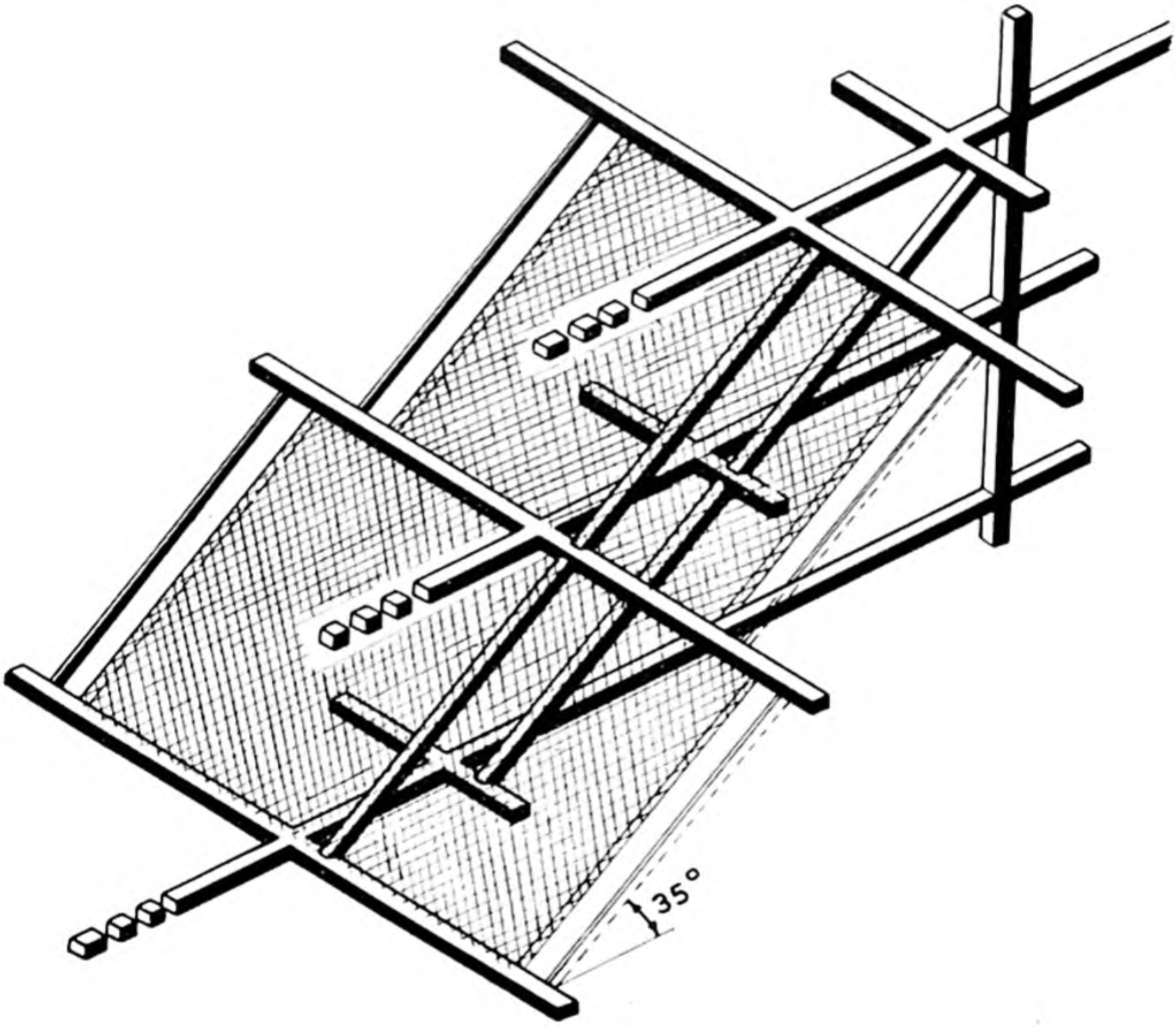


FIG. 27

is 85 yards, then the distance between horizons will be 170 yards. Extensive development is required in the strata between horizons in order to carry out this development, as indicated in Fig. 27, which shows a sub-level cross-measure drift and a staple shaft required in each district working several seams.

In steep measures dipping at about 40 degrees or more, the longwall method of working with a diagonal face layout is almost universally adopted. The length of the coal face in most cases is between

100 and 175 yards, which corresponds to a vertical interval of between 60 and 85 yards. It is then possible to fix the distance between horizons to a distance appropriate to the face length, and thus avoid intermediate sub-level development in the intermediate strata. Generally, main-level development is considered too expensive to repeat at intervals of 60 and 85 yards, as the life of each individual horizon is too short and the daily output is limited because the number of faces which can be worked simultaneously is reduced.

Thus, in steep measures a horizon interval which is at least twice, and very often is three or four times, the vertical height of the coal face has proved to be the most suitable distance. If this height is 60 yards, and the distance between horizons is 240 yards, four faces can be developed above each other in individual seams. A development in this manner is illustrated in Fig. 28. As well as the staple shaft to

A RETURN AIR HORIZON.
B. MAIN HAULAGE HORIZON.
C. SUBLEVEL CROSS MEASURE DRIFTS.

FIG. 28

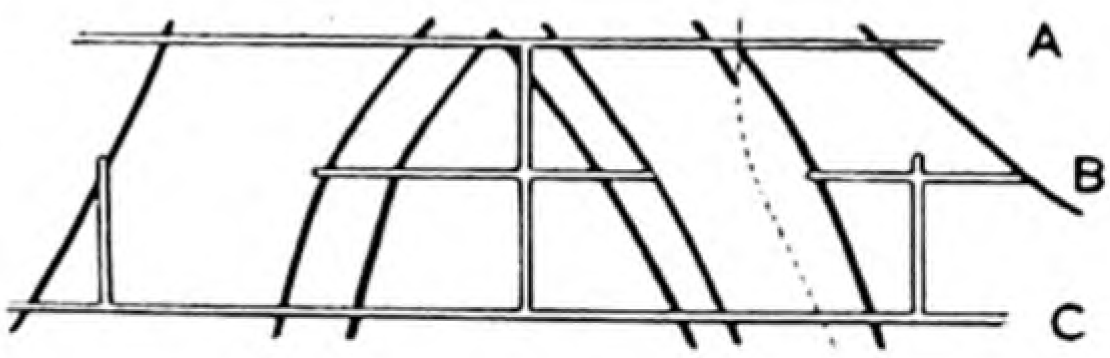
which reference was made in Section 4 (*c*), three sub-level cross-measure drifts are required in each district. Another example is shown in Fig. 29.

(*f*) *The distance between horizons in flat measures.* The distance between horizons depends to a great extent upon the intermediate reserves, as has already been discussed, but is not proportional to them. It is interesting to note that in collieries working flat seams in the Western European coal fields, especially in the Ruhr, the Aachen district, the Saar, Holland and in the Campine, the distance between horizons varies between 100 and 200 yards. The seam density in these fields ranges between 0·9 and 4 per cent. The horizon interval approaches the lower limit when the seam density is high, and the higher distance when it is low.

The coal reserves between two horizons in a colliery concession or 'take' of about 7×10^6 square yards, which is the average reserve

area of a colliery unit in the Ruhr or Aachen coal field, amount to from 10 to 25 million tons. The average annual output of such a colliery is about 1 million tons and the life of the main haulage horizon varies between ten and twenty-five years. These periods are considered to be the most favourable for the redemption of capital expenditure incurred in the development and maintenance of the horizon.

Considering the different 'seam densities' and the distance between horizons, a horizon in the lowest seam density region has the shortest life. Taking the horizon interval at 160 yards, which is the upper average previously mentioned, and the 'seam density' as 0·9 per cent., the reserves between horizons would be $0{\cdot}9 \times 1{\cdot}6 \times 7 \times 10^6 = 10 \times 10^6$ cubic yards. Taking the specific gravity at 1·3 and the saleable percen-



A. RETURN AIR HORIZON.
B. SUBLEVEL.
C. MAIN HAULAGE HORIZON.

FIG. 29

tage at 80, this corresponds to a reserve of about 8×10^6 tons of saleable coal. The life of a horizon in this case is consequently eight years. This period can be increased if the distance between horizons is increased, and it may considered, in such a case, that a distance of from 180 to 200 yards between horizons is justified.

On the other hand, a horizon in strata which has a high seam density has a longer life even if the distance from the return airway horizon is only 100 yards. The coal resources within this distance may, for example, be $2{\cdot}5 \times 1 \times 7 \times 10^6 = 17{\cdot}5 \times 10^6$ cubic yards or 17×10^6 tons. Taking 80 per cent. of this as saleable, the reserves would be about 14×10^6 saleable tons. The life of such a horizon would be about fourteen years, assuming an annual output of 1×10^6 tons. Due to the higher coal resources available, the development cost per ton of this horizon would, in this case, be less than in the previous example.

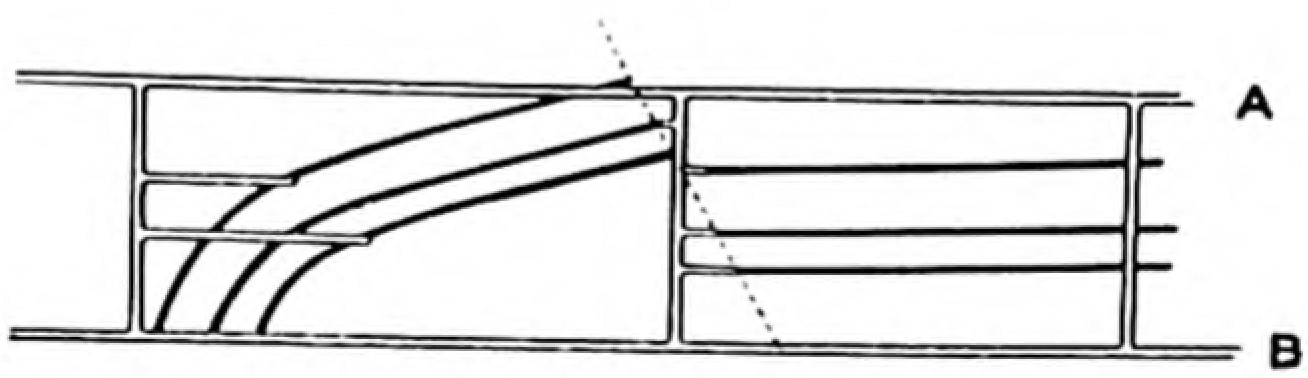
In the last twenty years the tendency has been to increase the dis-

tance between the horizons, and it is probable that this progress will continue within certain limits. This development is due mainly to the improvements in staple-shaft winding technique and roadway supports, with a consequent reduction in cost. The improvement in the system of roadway supports has resulted in a longer life in the main horizons than was formerly the case. The present-day policy of increasing the annual output of the colliery unit has also influenced the increase in horizon interval but, as all other conditions remain the same, this policy tends to reduce the life of the horizon.

Further details on the size of unit are discussed in Section 9.

(g) *The horizon interval in semi-steep and steep measures.* The distance between horizons in collieries of the Western European coal fields, working in semi-steep and steep measures, is governed by the factors previously discussed. The accepted distance is generally dependent upon an examination of the influence of the coal reserves and the length of face to be employed—that is, the difference in level between the ends of the coal faces. In semi-steep measures, distances of about 100 to 125 yards are common, whereas in steep measures intervals of from 150 to 240 yards are usual.

It frequently happens that in the same development area there is a variation in seam inclination from flat to semi-steep and steep. In such cases the choice of horizon interval must be a compromise, taking into consideration all the factors involved. Fig. 30 illustrates such a case.



A. RETURN AIR HORIZON.
B. MAIN HAULAGE HORIZON.
FIG. 30

Section 6. Development and Layout of the Coal Faces between the Horizons

(a) *General remarks.* The objective of the development in the strata is to prepare a network of roadways commencing from the main shafts with the cross-measure drifts, laterals and staple-shaft

connections, and dividing the colliery unit into districts which are available for the final extraction of the reserves. The coal-face development which follows within the districts is directed towards the preparation of the individual faces for coal-getting. The layout of the coal faces refers specifically to their length, direction of advance, distribution within the district and their relationship to the colliery as a whole. The roadways required for the development of the faces are driven in the seams, and their layout depends upon the system of

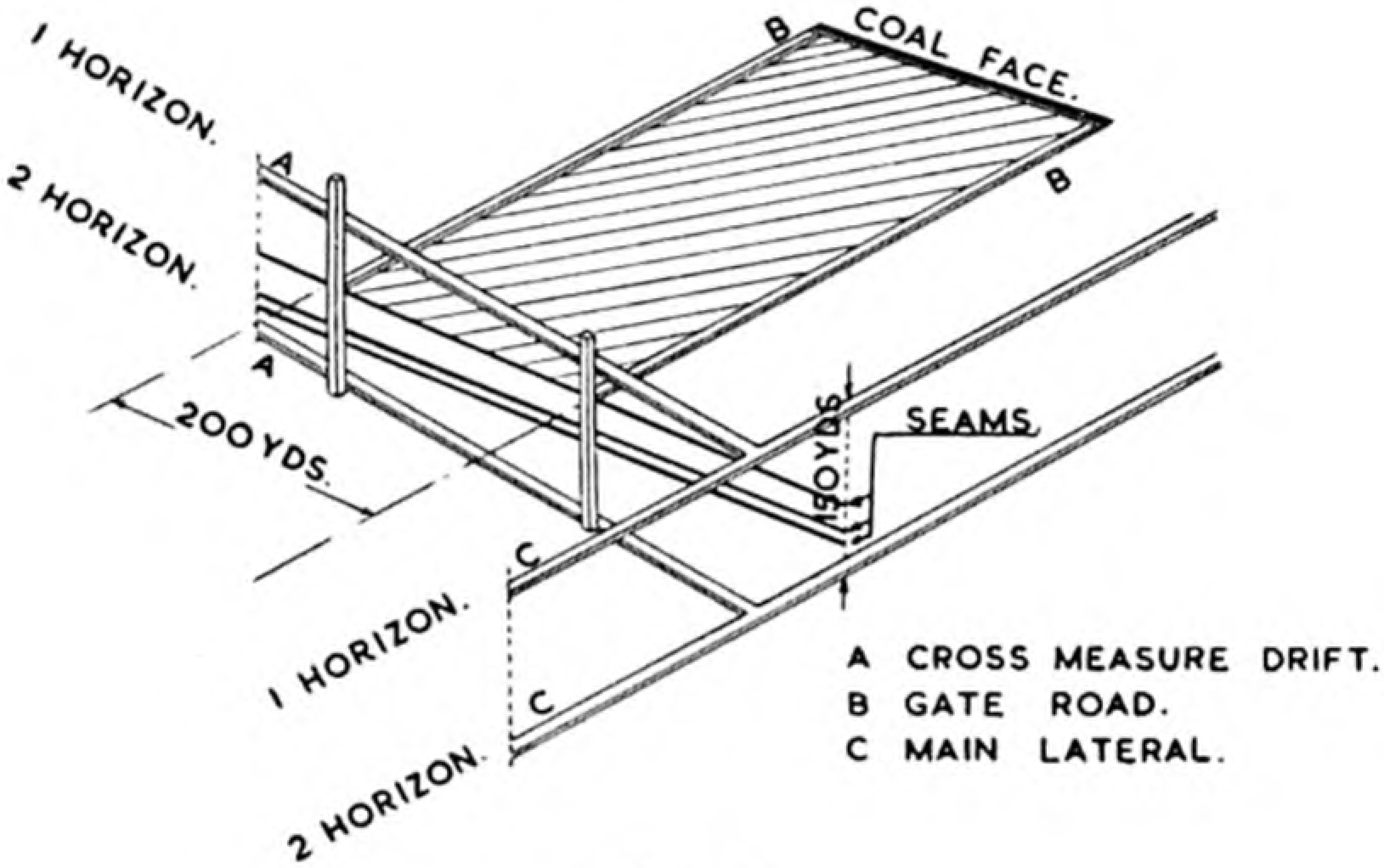


FIG. 31

working to be adopted. This system may be longwall advancing or retreating, either to the strike or to the dip, in flat, semi-steep or steep measures.

(*b*) *Coal-face development in flat measures.* In developing longwall faces advancing in the direction of the strike, two level gate roads are driven in the seam. The distance between these gate roads corresponds to the length of face to be won out. The gate roads are generally parallel to each other, but deviations in their direction may occur due to variations in the dip of the seam, although their direction may be maintained at the expense of change in gradient, especially if gate conveyors are used. If single-unit faces are being developed, the upper gate or lower gate may be used for coal transport or, as is normally the case, coal transport is confined to the lower gate as in Fig. 31.

These gate roads commence from the corresponding staple shafts.

During the course of face development, the gate roads are driven about ten yards on either or both sides of the shaft, as illustrated in Fig. 32, and the face is developed by connecting the gate roads with a 'rise' or holing across from either, or both, the rise and dip side gate roads. On completion of this work, *vide* Fig. 33, the coal face is prepared for the installation of the conveyors and other face machinery, the gate roads being continued as the face is advanced, as in Fig. 34.

Both faces may be won out at the same time, using either solid stowing, strip-packing, or full caving methods. On the other hand, a single face may be won out followed by the second face in the other direction, the rise in this case being maintained until the second face is due to start. With retreating longwall, the face development is delayed until the gate roads are advanced to the required limit, and the same method of winning out the face is adopted, *vide* Figs. 35, 36 and 37.

The retreating longwall system has the advantage of proving the seam between the gate roads, before the face is commenced, and allowing adequate steps to be taken to meet any difficulties. The main disadvantage of the system is the delay incurred before extraction can take place, and the necessity to transport gateway ripping material outbye, whereas in most cases, in the advancing system, it can be stowed immediately in the waste. Because of these advantages, advancing longwall is almost universally used in the Western European coal fields. Faces to the rise or to the dip may also be developed by either the advancing or the retreating system, the face development under these conditions being quite different. A typical development to the rise is illustrated in Fig. 38. The lower gate road must be driven to its full extent, and the upper gate road mid-way along the striking length to be worked. A rise is then driven in the seam from a point mid-way along the lower gate road to connect with the end of the upper gate road. This rise serves for the return ventilation from the faces, and for the transport of stowage material and supplies to the faces which advance to the rise as a double unit.

Another example is illustrated in Fig. 39, in which the development work prior to extraction is even more extensive. Two gate roads, an upper and lower, are driven to the full extent of the panel and are connected by two rises, one at each end. These rises serve for ventilation and transport of material. The face is won out between the rises which act as tailgates to the double-unit face, and the centregate,

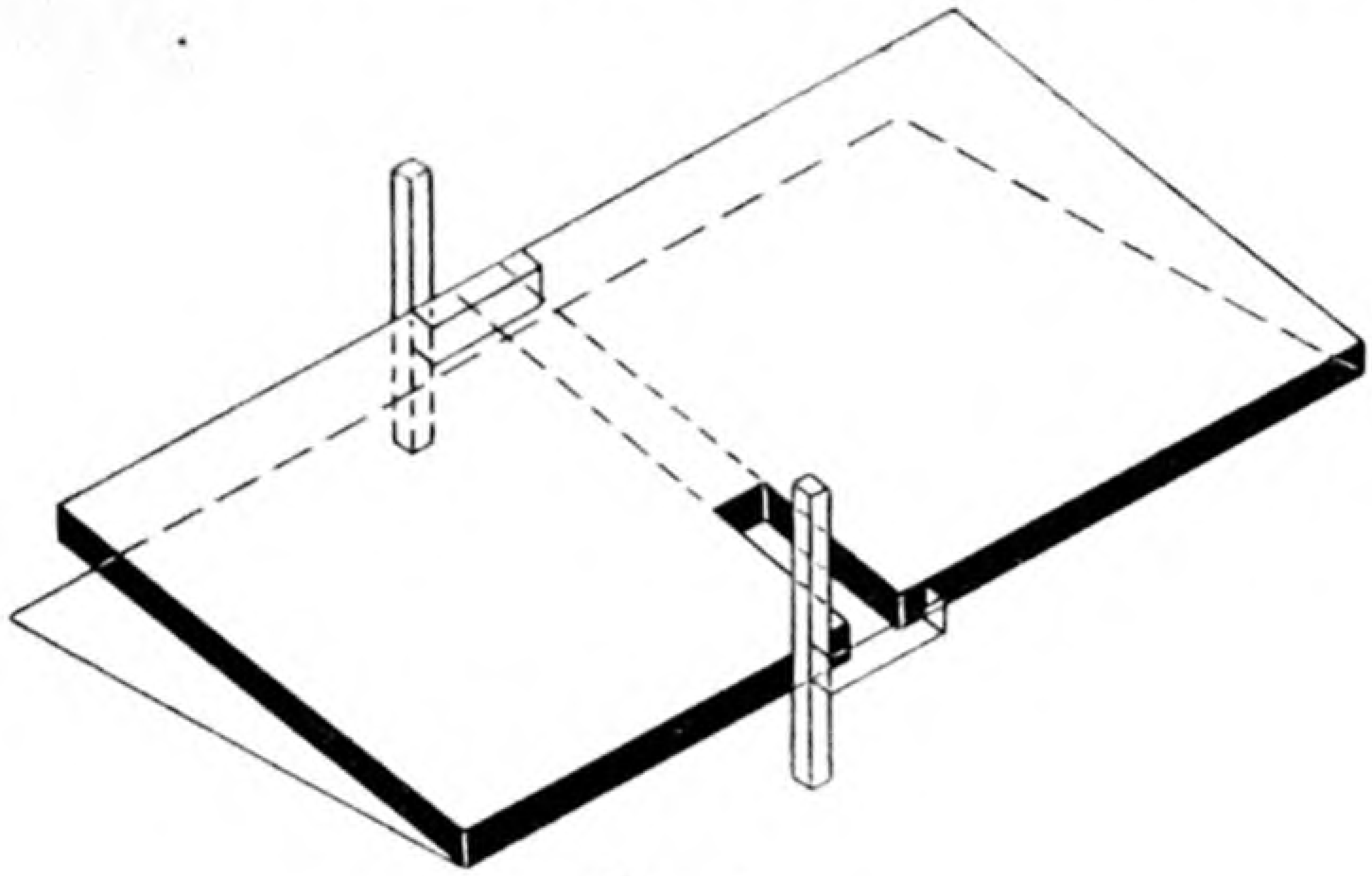


Fig. 32

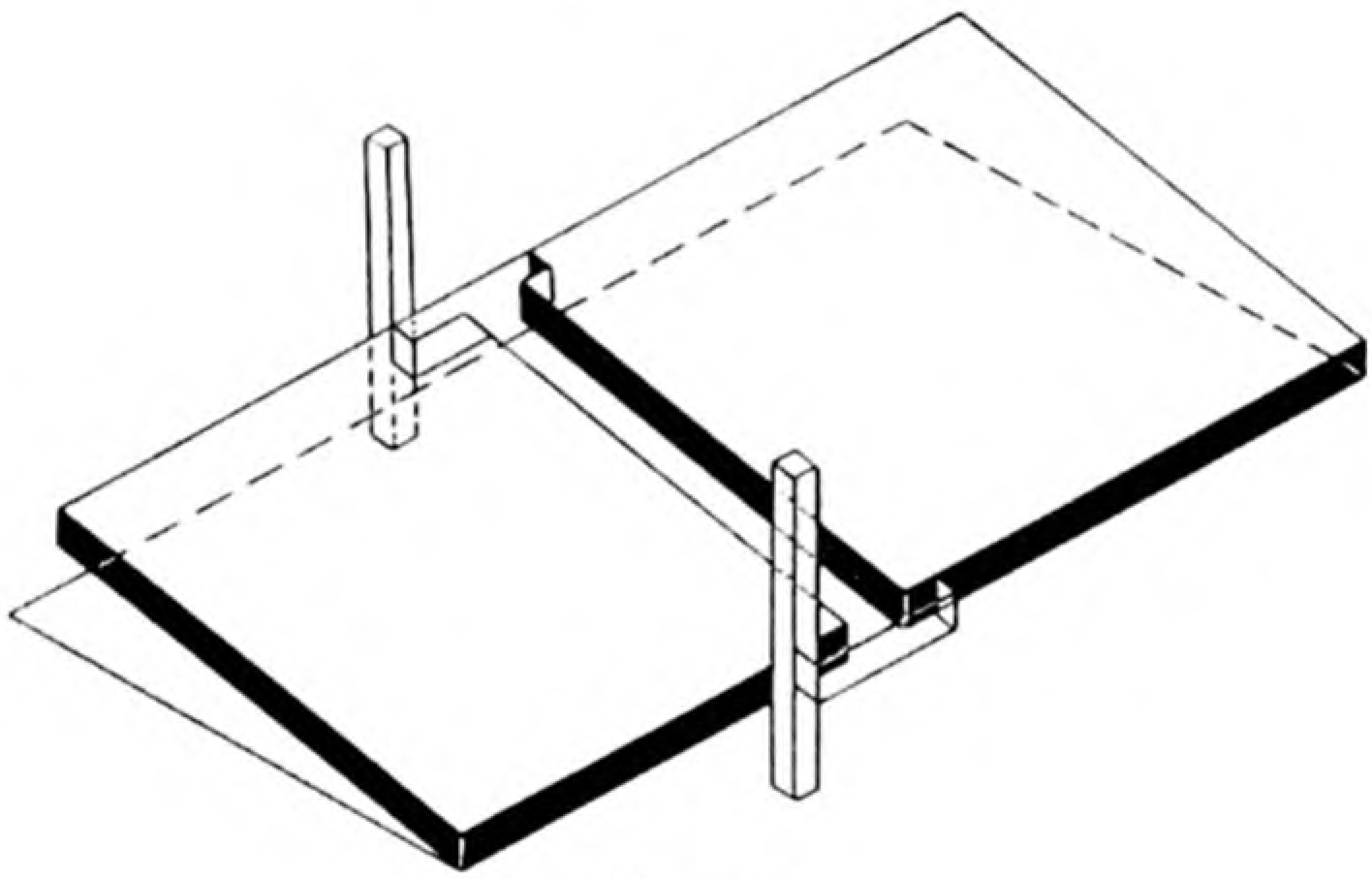


Fig. 33

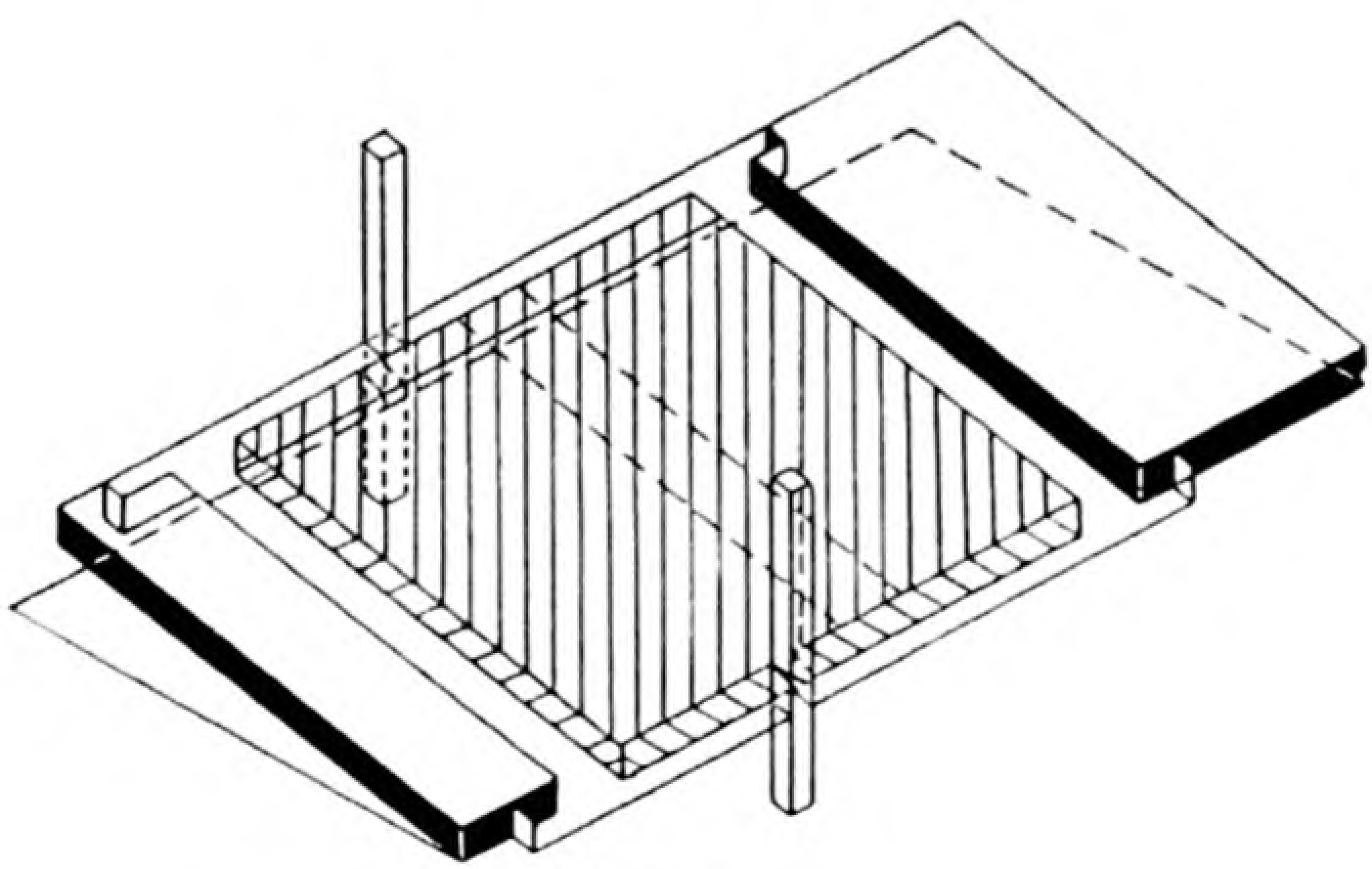


Fig. 34

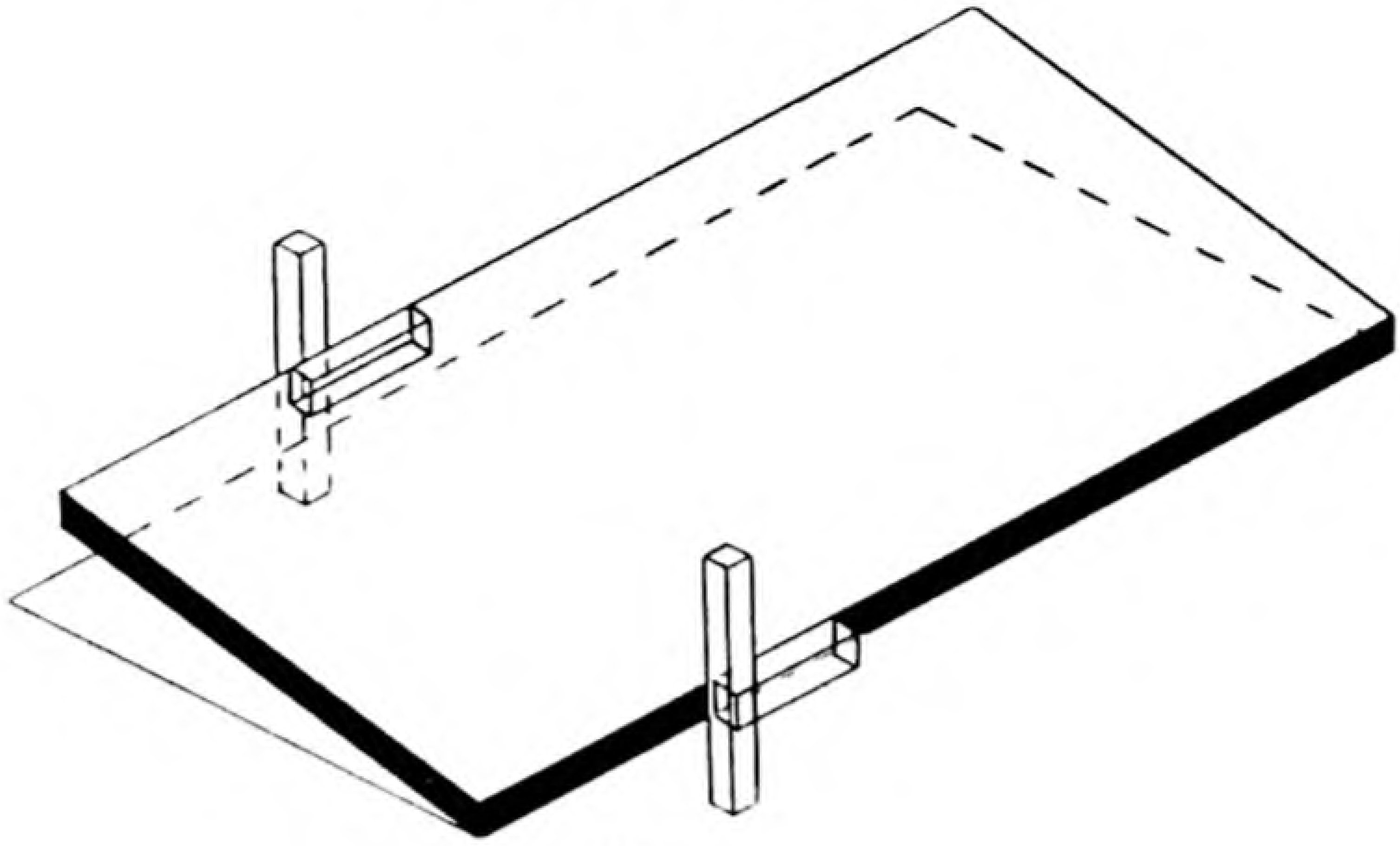


Fig. 35

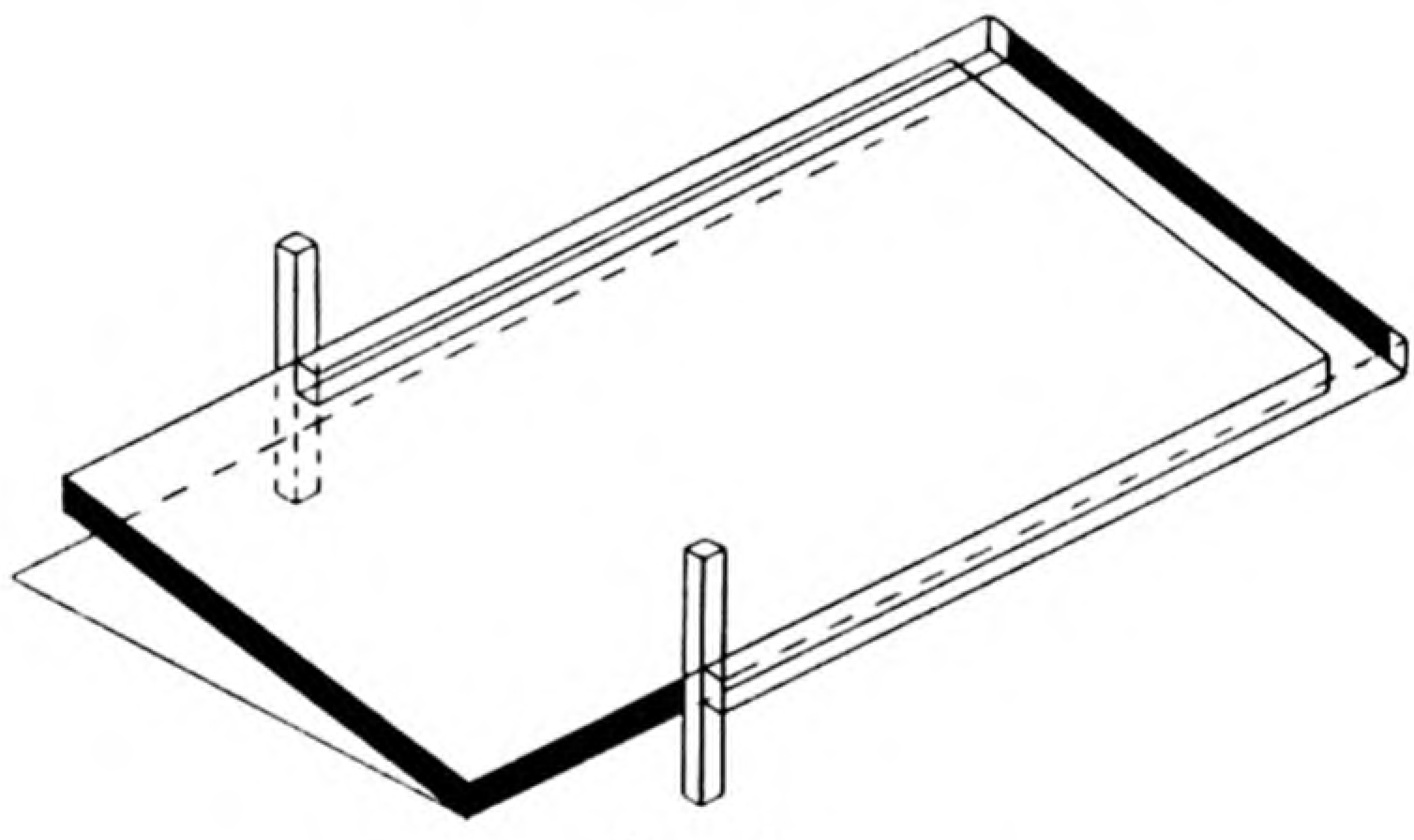


Fig. 36

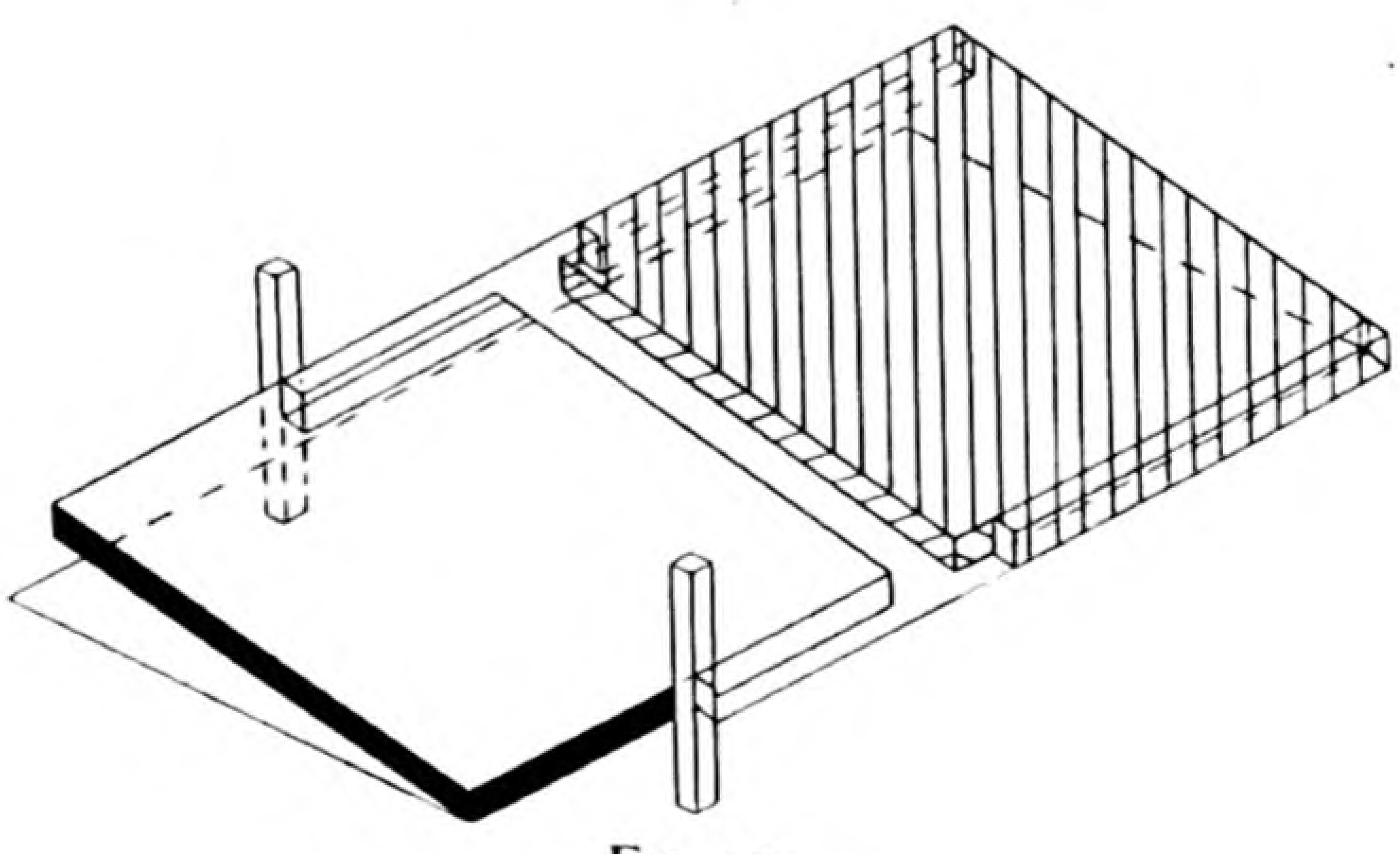


Fig. 37

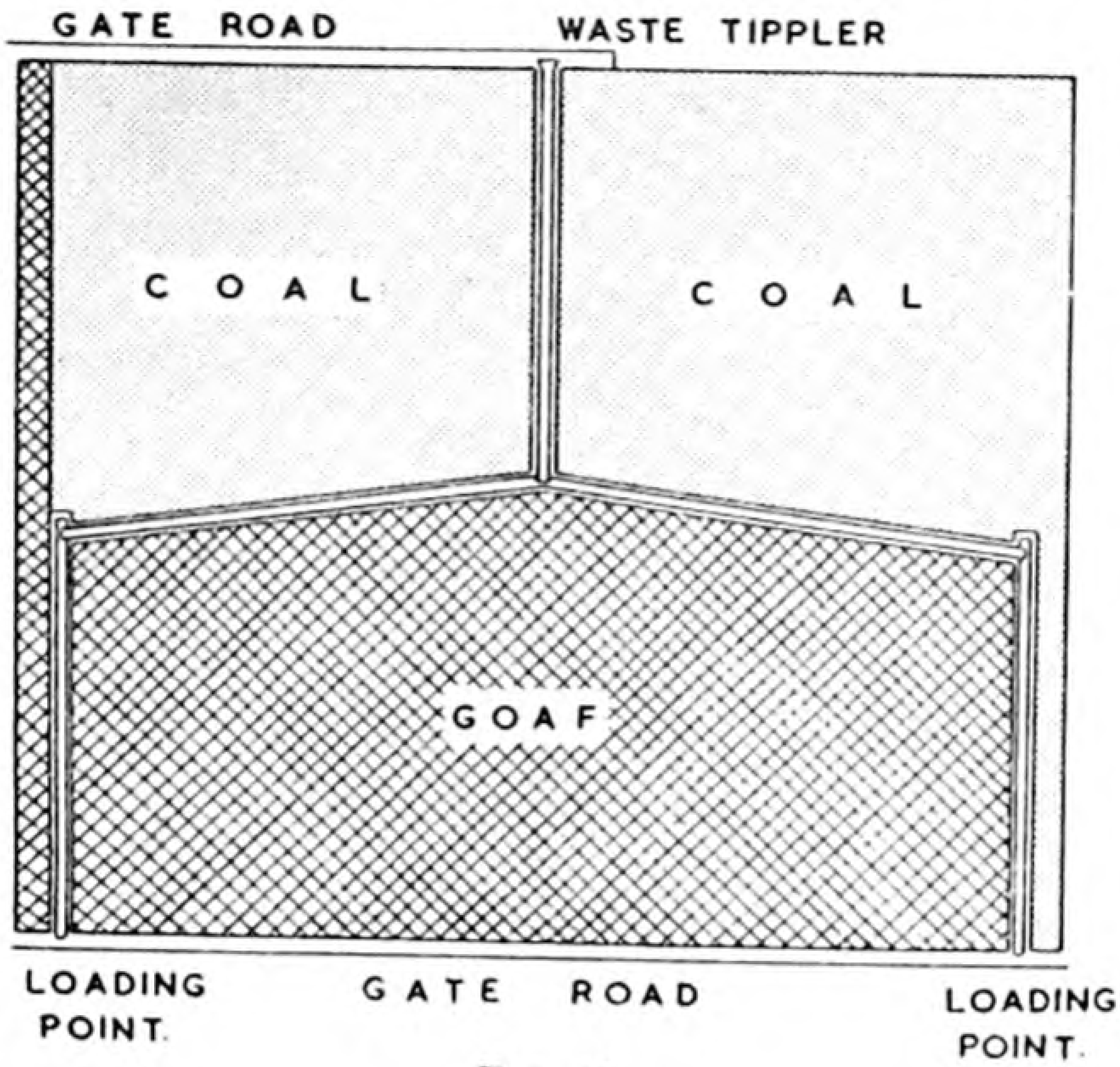


FIG. 38

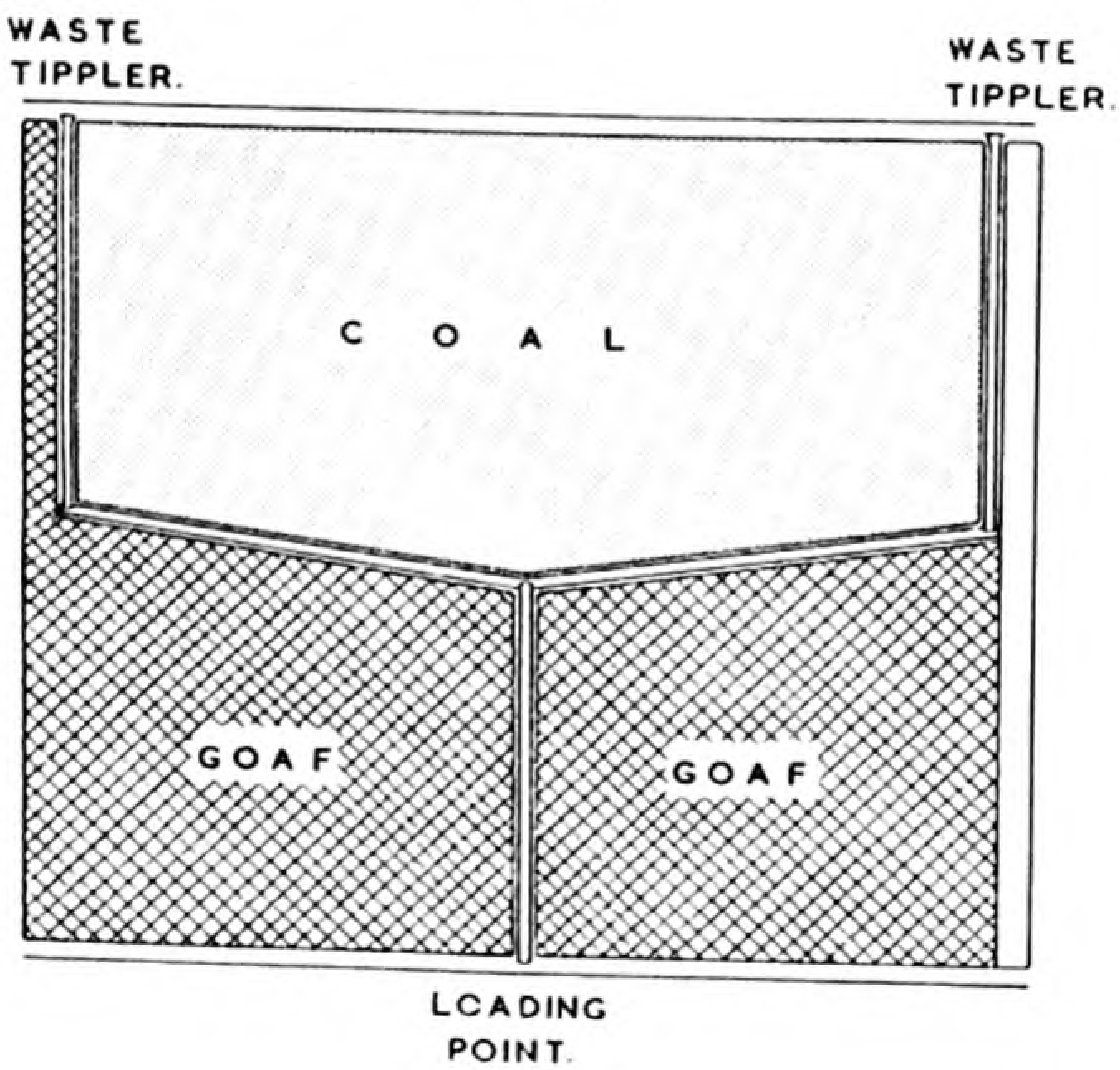


FIG. 39

along which the coal is transported, is developed as the faces advance and acts as the intake to the faces.

In comparing these methods, it should be noted that in the former case there are two loading-points, one at each end of the panel, and

one central tipping-point for stowage material. In the latter case, only one central loading-point is used, and two tipping-points, one at each end of the panel, are required. The main advantage of this system of working to the rise lies in the fact that stationary loading- and tipping-points are employed during the life of the panel, whereas in the development of faces to the strike, the loading- or transfer-points are advanced daily with the faces.

A further system of development to the rise, which may be convenient for the development of short longwall faces, is shown in Fig.

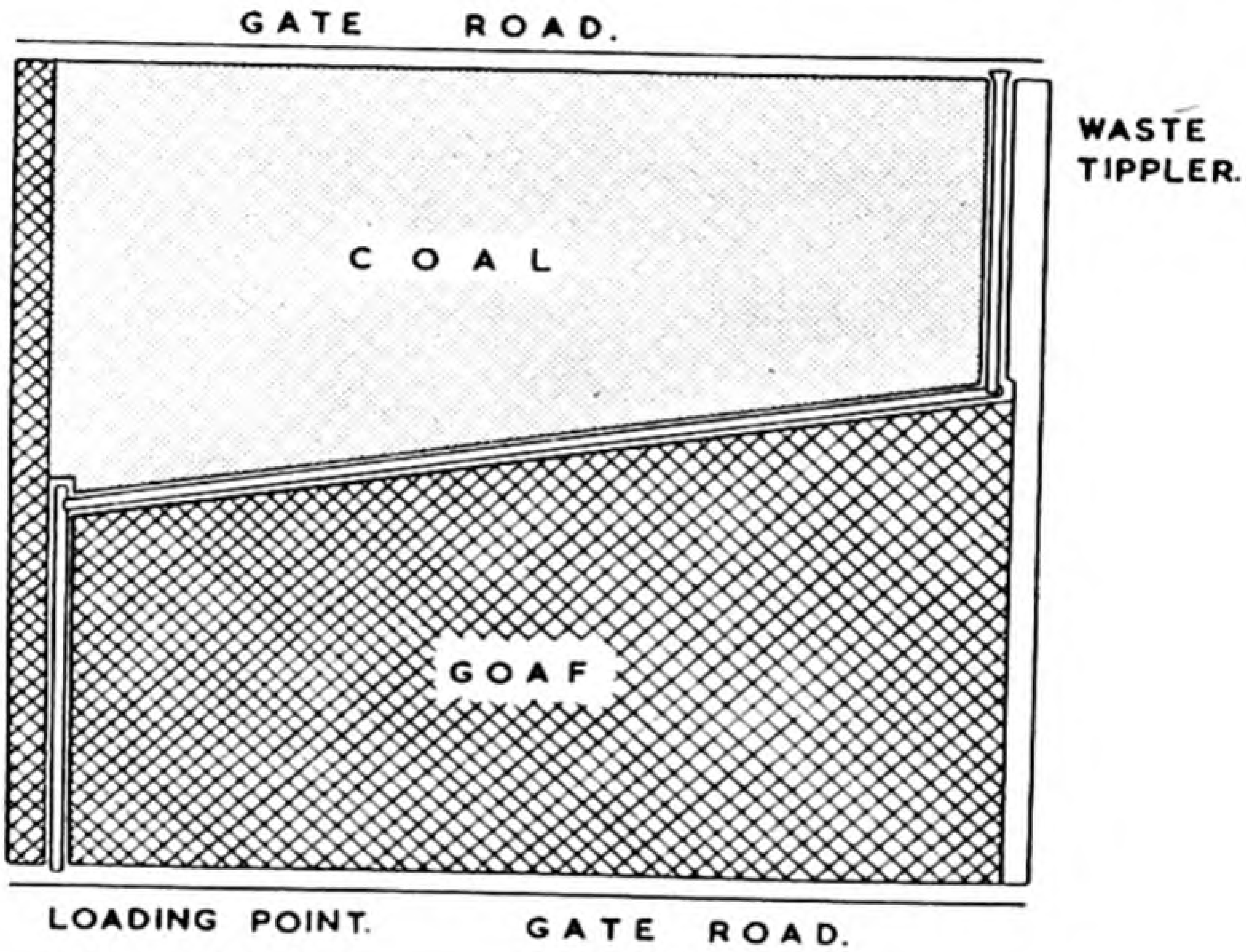


FIG. 40

40. In this case a gate road to the strike is driven the full length of the panel, either leaving a pillar on the upper side of a lower sub-level, or the lower horizon, or immediately above this level. A rise connecting the lower and upper levels is driven at the extreme end of the panel, and a face is developed to the rise from the striking gate road. Further details of this method of working are discussed in Chapter 4, Part III. In developing faces to the dip, similar systems can be adopted. In working to the dip or to the rise, consideration must be given to the limits of conveying on the face and in the gates.

(*c*) *Face development in semi-steep and steep seams.* The development of faces in semi-steep seams is identical with the methods used in winning out in flat measures, with the exception that working to

the dip or to the rise is not to be recommended, while faces to the strike are the most common. The face line is developed from a rise in the seam when adopting advancing longwall. In retreating longwall, the gate roads are driven the full length of the panel and connected by rises at each end.

In steep-seam face development, the face is gradually won out on a diagonal face line with the bottom end of the face leading. The face line can be either 'stepped' or 'saw blade', as shown in Figs. 41 and 42, or on a 'buttock' system. The gate roads in steep seams are not started

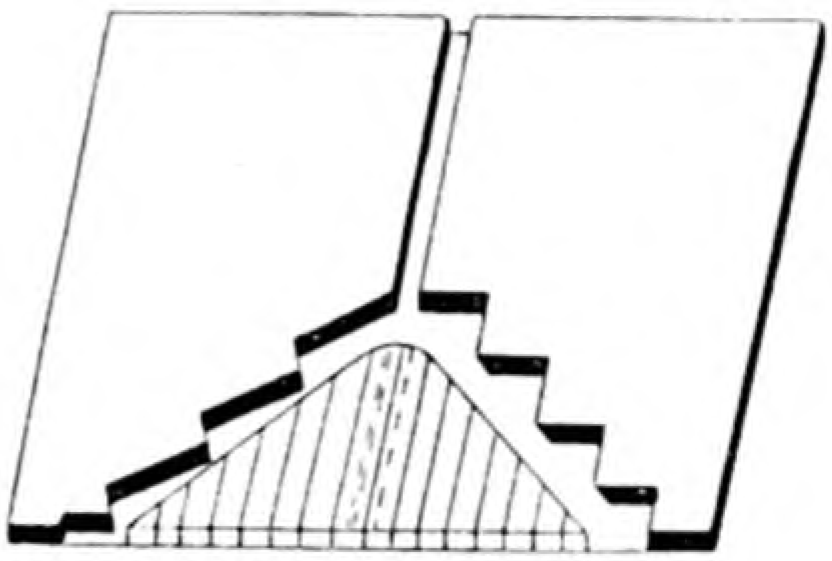


FIG. 41

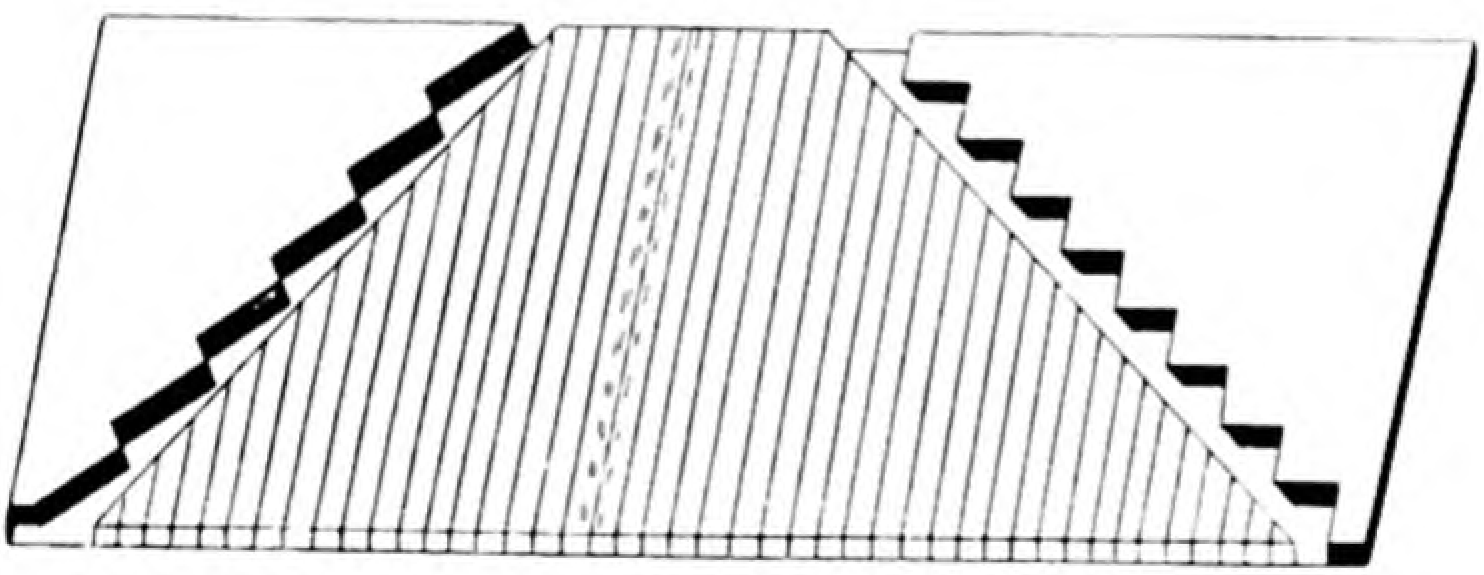


FIG. 42

from the staple shafts, as in flat-seam face development, but from sub-level cross-cuts driven from the staple shafts. Refer to Fig. 19, Section 4 (*c*).

(*d*) *Layout of longwall faces in a district in flat measures.* In the horizon mining system a district is limited in depth between the district cross-measure drift on the main haulage level and the corresponding cross-measure drift at the return airway level, and in the lateral direction by the staple shafts connecting the two horizons. Within this unit the faces developed represent a single production unit which is also a separate ventilation split.

The distribution and organisation of the individual faces in this

production unit can now be discussed. It is usual to find that the highest seams are worked first and extraction progresses to the lower horizon in sequence. This rule has few exceptions in flat measures, but in steep-seam formations deviations from this principle are frequent. Fig. 43 shows a cross-section through a district unit in which there are three seams present, averaging from 3 to 4 feet in thickness, lying between two horizons which are 110 yards apart, while there are no major faults present and the average dip is between 1 degree and 5 degrees.

The length of the coal faces must be decided, since the number of staple shafts and the distance between them are based upon the actual length chosen. Full caving is to be applied and, according to experience in working similar seams in other districts at the colliery, the most reasonable length is considered to be 250 yards. The daily output per coal face, taking a rate of advance of 5 feet in a seam 4 feet

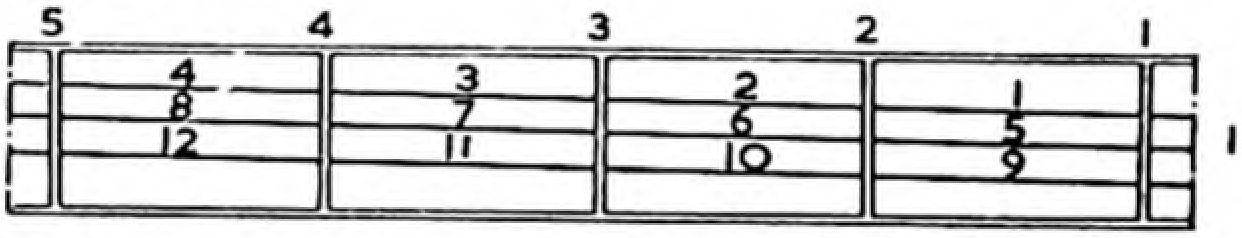


FIG. 43

thick, will be about 400 tons of saleable coal, and in a 3-feet seam about 300 tons. Taking a working panel length in the direction of strike of 600 yards, the life of each pair of faces will be 1,800/5 or 360 working days, or about $14\frac{1}{2}$ months (taking 250 working days per annum).

The first pair of faces will be developed in the upper seam and worked in both directions to the strike between staple shafts 1 and 2. While these faces are being worked, another pair of faces will be developed in the same seam between shafts 2 and 3, and the development of pairs of faces is continued in sequence between shafts 3 and 4 and between shafts 4 and 5. The middle and lower seams are worked in sequence in the same manner, working being confined to one seam at a time, as shown in Fig. 43. This sequence of working implies the provision of all district staple shafts in advance of extraction of the lower seams. These shafts must be fully equipped, but only one shaft is in use for winding coal and one for return air and supplies at any period of working. For instance, before the fourth pair of faces are developed, all staple shafts must be driven and ready for use, and

when these faces are worked, only shaft 4 is used for transporting coal, while the upper part of shaft 5 is used as a return airway.

The output from this system of development is restricted to the tonnage from one pair of diverging single-unit faces, each 250 yards in length—that is, say, 800 tons (saleable). Half this output would be obtained if only one single unit is worked at a time within a district. In the former case, the life of the district at 800 tons per day would be $12 \times 14\frac{1}{2}$, or 174 months, or $14\frac{1}{2}$ years.

The disadvantage of forward staple-shaft development can be overcome in several ways, as shown in Figs. 44 to 47, in which the sequence of working the individual seams between staple-shaft positions is illustrated.

The first method shown in Fig. 44 consists in working double-unit faces in each seam in descending sequence between shafts 1, 2 and 3, and in continuing this development between shafts 3, 4 and 5.

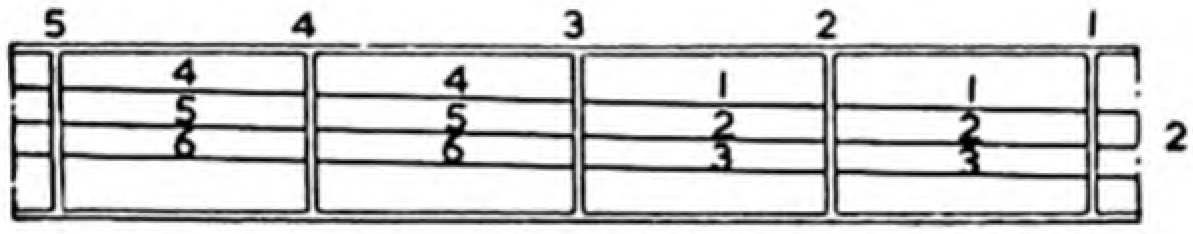


FIG. 44

The double-unit faces may be developed either singly or as 'diverging' faces on each side of the staple shaft or cross-measure drift. In the first case, the output per day would be the same as before, i.e. 800 tons per day. Where diverging faces are developed, the output is doubled to 1,600 tons per day. The life of the district at 800 tons per day is 174 months, and 87 months at the double output. In this case the sinking of shafts 4 and 5 can be delayed until they are finally required, either just before 87 or $43\frac{1}{2}$ months after the commencement of extraction in the district. With the double-unit system of development there is also a higher efficiency in the concentration of output to the loading-points at staple shafts 2 and 4.

The ventilation system will include staple shaft 2 as the intake to the faces, with the return air travelling up the upper parts of shafts 1 and 3 to the return airway horizon. Another variation can be used if intake air is brought up shafts 1 and 2, the intake from shaft 2 acting as an additional intake to the faces. The same system is repeated between shafts 3, 4 and 5.

A further variation in the development of double-unit faces is

shown in Fig. 45. In this instance the single-unit face development shown in Fig. 43 is replaced by a double-unit face development, with a consequent doubling of the daily output and reduction of the district life by half. The ventilation system employed in this case is similar to that used in the system discussed with reference to the development shown in Fig. 43. Comparing the schedules of extraction illustrated in Figs. 44 and 45, consideration of surface subsidence may influence the decision to work in either manner. Where gradual

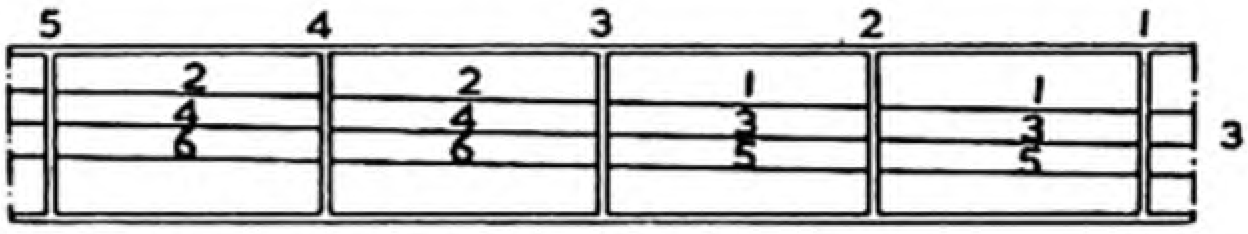


Fig. 45

subsidence over a larger area is considered better in the circumstances, the system shown in Fig. 45 would be an advantage. Further consideration is given to this question in Chapter 2.

The variation shown in Fig. 46 is an admirable one, if a concentration of output from the three seams to one loading-point is required. Each staple shaft is used to its highest capacity and the successive blocks between staple shafts are worked out in sequence. This system has the disadvantage of concentrating working within a comparatively small area, with possible major repercussions at the surface. The ventilation system is similar to the case discussed for the

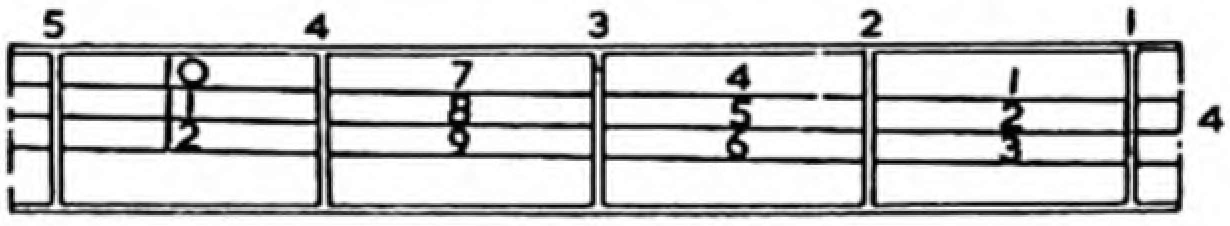


Fig. 46

system shown in Fig. 44. Single-unit faces are being developed and, since these can be worn out singly or in diverging pairs, the output per day is 400 or 800 tons, and the life of the district 174 or 87 months.

The system of development shown in Fig. 47 illustrates a method of single-unit development whereby the output can be raised to the same figure as with double-unit faces. This is done by staggering simultaneous, single-unit development between staple shafts 1 and 2 and shafts 3 and 4, and working the three seams in sequence within these blocks in depth. The ventilation system is the same as for normal single-unit development. The output per day from two

single-unit faces will be 800 tons, and from two pairs of diverging single-unit faces 1,600 tons. The life of the district will therefore be either 87 or $43\frac{1}{2}$ months.

The disadvantages of the system from the point of view of surface damage is the same as in the last case. Subsidence can be minimised if

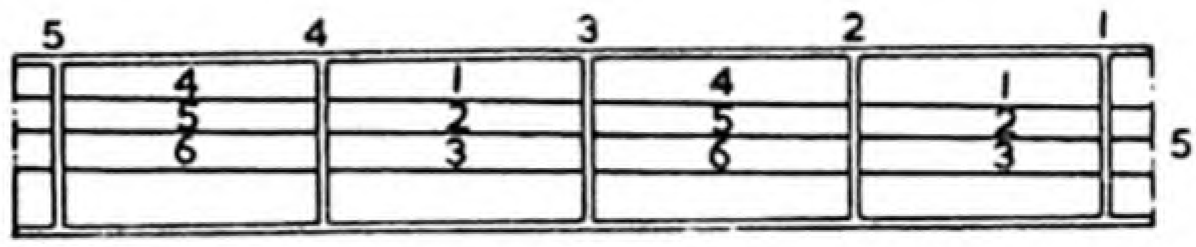


Fig. 47

the sequence of working is organised to take place in each seam in turn between staple shafts 1 and 2 and between 3 and 4, followed by working between 2 and 3 and between 4 and 5, always in the same seam.

(*e*) *The layout of coal faces in a group of seams, using one staple shaft.* A method of developing and working a group of seams in which the

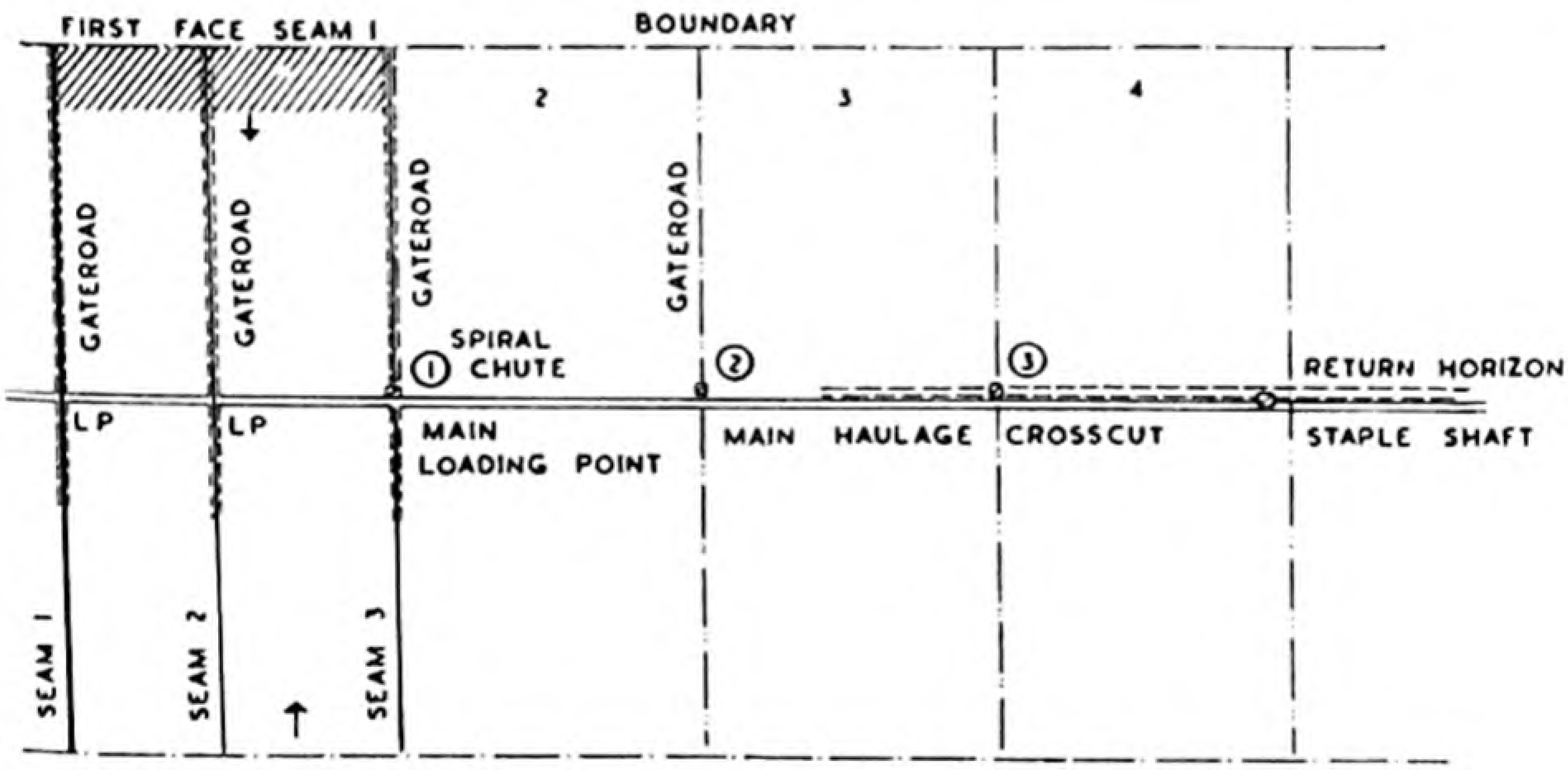


Fig. 48

main characteristic is the use of only one staple shaft is illustrated in Fig. 48. The normal series of staple shafts is replaced by a number of short vertical shafts of small cross-section equipped with spiral chutes. The cross-section in Fig. 49 shows a group of three semi-flat seams, in which seam 1 is 7–8 feet thick and seams 2 and 3 are 4 feet and 6 feet thick respectively.

The single staple shaft connecting the upper and lower horizons is

situated at the highest point of the seams and serves as return airway to that horizon as well as for transporting materials and the men working on faces near to the shaft. The method of working employed can be either advancing or retreating longwall. In the case considered, retreating faces are being worked. In order to develop the faces in each seam, a gate road and a rise are required. The gate roads are driven from the district cross-measure drift to the limit of the panel to be won out. Roadways are driven to the rise connecting the gate roads driven from the short spiral chute shafts and at the limit of the panel, first in the upper seam and afterwards in the middle and lower seams. These rises serve as return airways leading towards the staple shaft and for transporting material brought through the

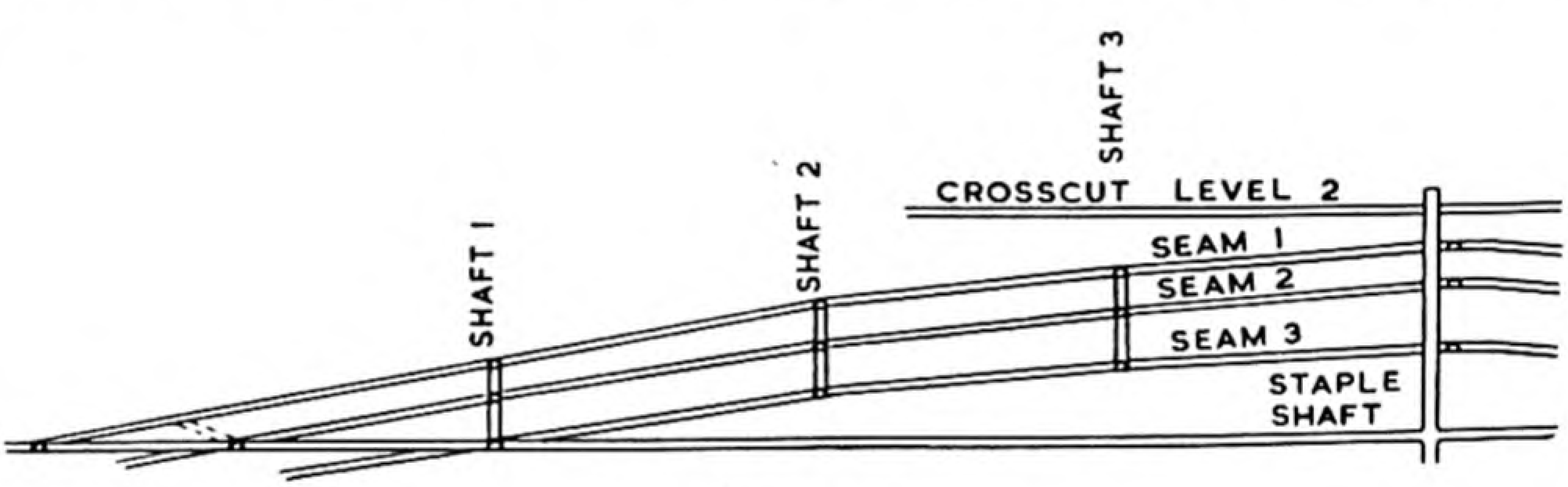


FIG. 49

staple shaft from the main haulage horizon. The coal faces are won out from the rises, the length of the faces being about 800 yards in seam 1, 725 yards in seam 2, and 600 yards in seam 3. These lengths are sub-divided into four coal faces in seams 1 and 2, and into three faces in seam 3.

The first face to be worked is the lower face in seam 1, the two middle and upper coal faces follow in sequence. During the retreating of the lower face in seam 1, another rise is driven in seam 3 in the vertical plane of the district cross-measure drift, which is the finishing line of the faces retreating from the boundary. This rise serves as the main transport road for faces 2, 3 and 4 of seams 1 and 2, and the faces 2 and 3 in seam 3, *vide* Fig. 48. The vertical connection between the gate roads in the upper seams and this rise is made by the small vertical shafts equipped with a spiral chute. These chutes replace the staple shafts normally required at the same intervals. In this instance they are about 60 yards shorter in length than the equivalent staple

shaft, and it is not necessary to install winding equipment. The cross-section of the shaft required is therefore less than would be needed if a normal staple shaft were used, in which space would be needed for cage, counterweight and ladder compartments. An additional ad-

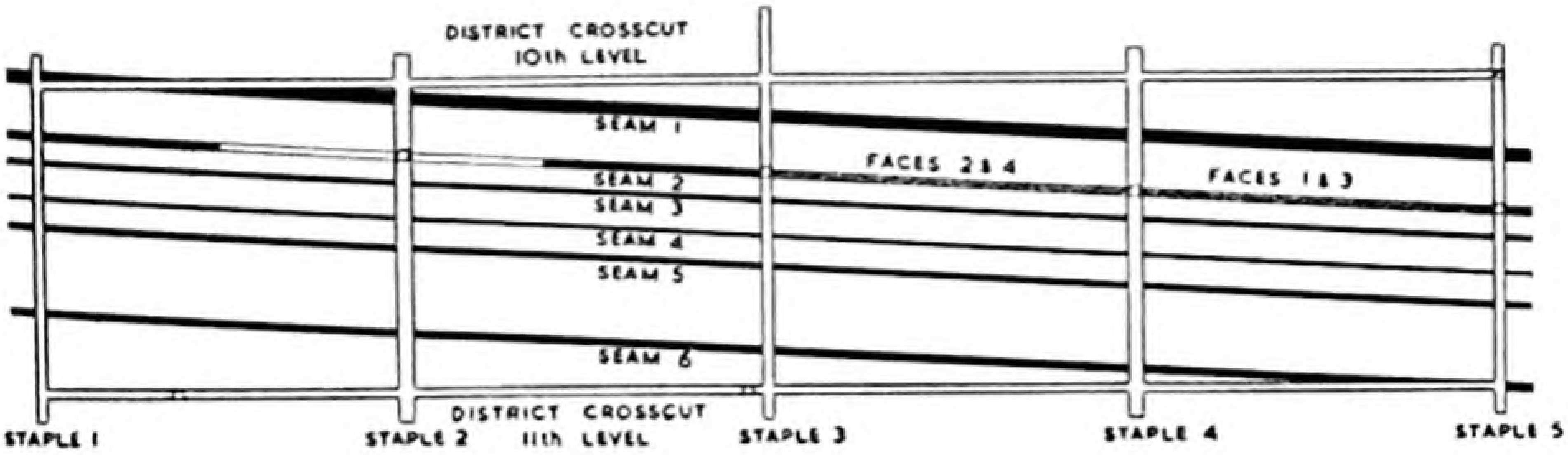


FIG. 50

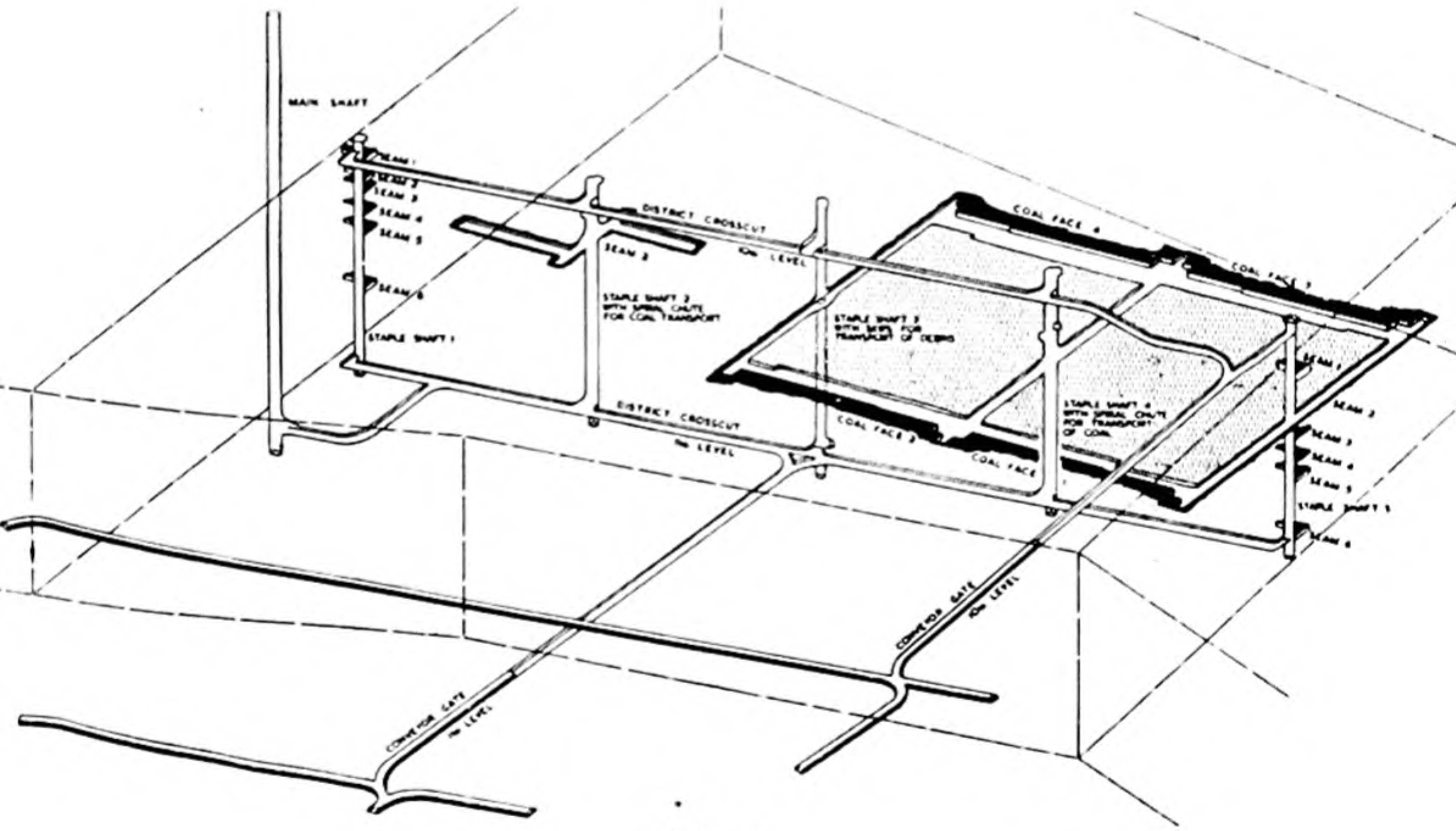


FIG. 51

vantage of this system of layout is the long life of the single loading-point at spiral chute shaft 1 in the main haulage horizon. In the zone lying between the main haulage horizon and spiral chute shaft 1, the coal may be loaded at the main haulage horizon from the faces in seams 1, 2 and 3, or conveyed to the rise in each instance to spiral

chute shaft 1, thus maintaining the single loading-point for the whole output from the district. On the other hand, the output from face 1 in seam 2 can be conveyed via a short rise drift in stone to the transport rise in seam 1 and loaded at the same point as the output from face 1, seam 1, thus eliminating the middle loading-point for seam 2 on the main haulage horizon.

The following example illustrates an interesting layout of a group of five seams. Fig. 50 is a cross-section showing the five seams between two horizons with five staple shafts. The whole length of the district of 800 yards is sub-divided into four sections of 200 yards, corresponding to eight faces which will be worked as four double-unit faces, each 400 yards in length, and consequently there will be double-unit faces between staple shafts 1 and 3, and 3 and 5. Fig. 51 shows the layout of the district as a whole and the working of two double-unit faces between staple shafts 3 and 5 in the upper seam. The faces are diverging, one of the double-unit faces advancing to the east and the other to the west, direction of strike being east to west. Staple shafts 2 and 4 will serve as transport shafts for the coal from the various seam levels to the haulage horizon, beginning with seam 1 and working downwards in sequence. For this purpose a spiral chute is installed reaching from seam 1 to the main haulage horizon, *vide* Fig. 50. The upper parts of staple shafts 2 and 4, between the return airway horizon and seam 1, and the subsequent seams as extraction is continued downwards in each seam, serve for the transport of waste for pneumatic stowage. A tube is installed in one compartment acting as a hopper or bunker and feeding the two pneumatic machines at the lower level. When seam 1 is extracted, the spiral chute is taken out in the upper section of the shaft and the bunker tube increased in length to the next seam being worked and in which the stowage machines are now installed. This procedure is repeated as each seam is extracted. Staple shaft 3 is used exclusively for the transport of stowage material by means of skips from the haulage horizon to the return airway horizon, along which this material is conveyed by belt conveyors to staple shafts 2 and 4. Staple shafts 1 and 5 are ventilation shafts and serve at the same time for the transport of men and supplies. Coal-getting is carried out during the morning shift on one double unit and during the afternoon shift on the opposite unit. Stowing requires 12 hours, or $1\frac{1}{2}$ shifts, to complete. Since the total face length is about 800 yards, with a 4 feet

6 inches daily advance, in a seam 3 feet thick, the daily output is about 1,200 tons. This output can be maintained from the two double units for two years, which is increased to four years for the complete extraction of seam 1, giving a total district life in the five seams of twenty years at a daily output of 1,200 tons.

(*f*) *Layout and time schedule of production in flat seams.* The layout to be planned is in five seams lying between two horizons as illustrated in Fig. 52, which shows a cross-section through the district. The cross-measure drift at the return airway horizon is already in existence, as well as staple shafts 1, 2 and 3 and the cross-measure drift at the main haulage level between the main lateral road and staple shaft 3. Sections 1 and 2 have been worked out with the exception

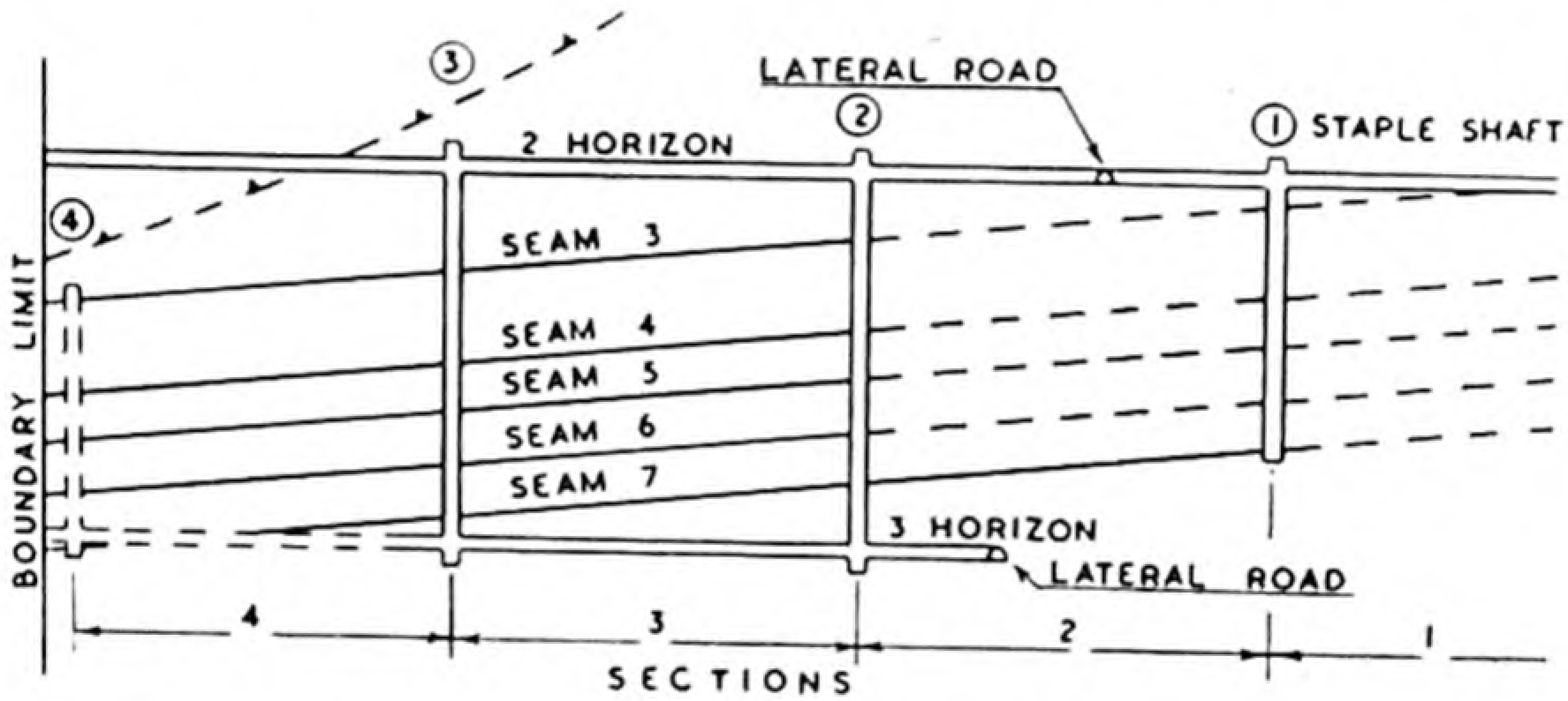


FIG. 52

of seam 7 in section 2. A daily district output of from 700 to 1,000 tons has to be maintained. This output can be obtained by working two single-unit faces simultaneously. In order to meet market requirements, it is preferable to have the two faces in different seams, in such a way that both faces advance to the strike but from opposite directions. In this way the eventual effects of strata pressure on either face can be avoided. Fig. 53 shows a plan of the district, indicating the different sections, the limits of the district development and the extent of the existing resources.

Fig. 54 illustrates the planned production chart for section 3 and includes the working of seam 7 in section 2 and of seam 3 in section 4. This sequence of production development is continued for the other seams in section 4. From the chart it will be seen that the first face to be worked in section 3 is to the west in seam 3. This face is

worked simultaneously with a face advancing to the east in seam 7, section 2. The development in section 3 is continued with an eastern face worked in seam 3, with a western face in seam 4, and so on. The daily output varies mainly according to the thickness of the seams, the output per manshift, and other seam and working conditions. The corresponding figures are shown on the chart, together with the manshifts required in the district to obtain the daily output target.

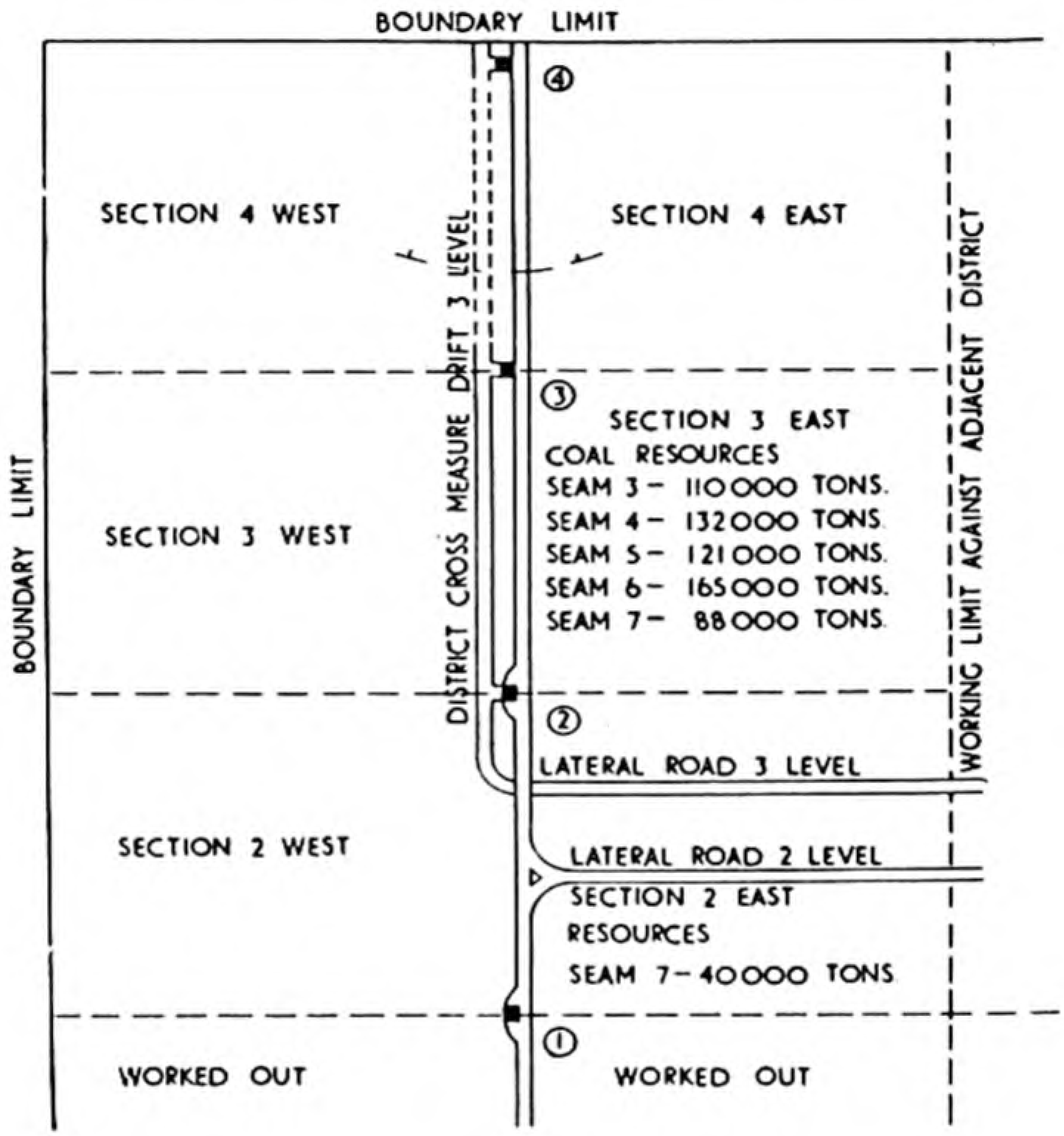


FIG. 53

The development in coal consists in winning out the gate roads and corresponding rises in each seam. These must be completed before coal extraction can begin.

The development in rock, including the driving of the cross-measure drift at the main haulage level from staple shaft 3 to staple shaft 4, and the sinking of staple shaft 4 upwards, must be finished before the face in seam 3, section 4 west can commence. Other examples of planned development and production charts from British mining practice are shown in Figs. 55 and 56.

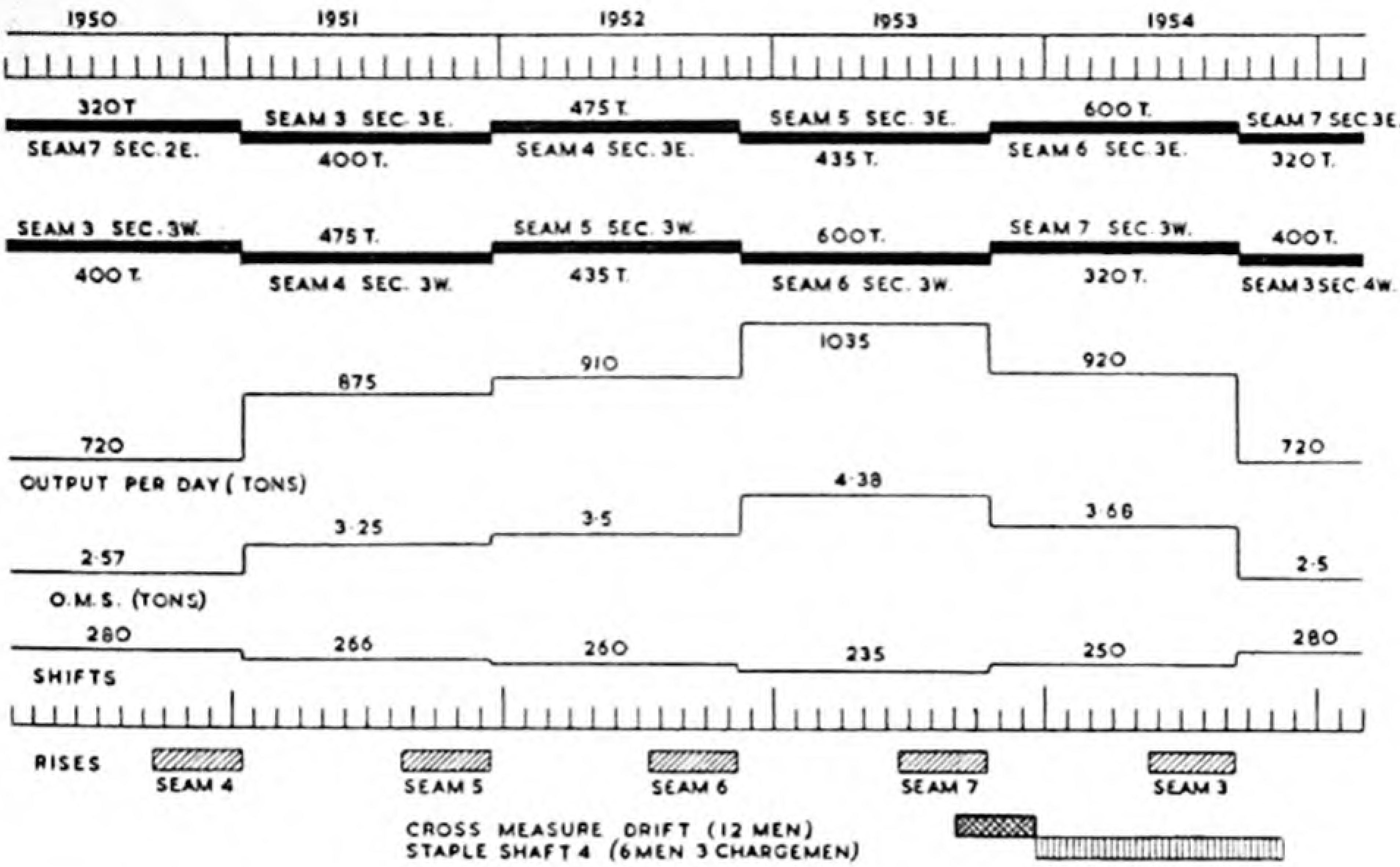


FIG. 54.

(*g*) *Layout and time schedule of production in semi-steep seams.* The layout of faces in semi-steep seams is quite different from that in flat formations. The coal faces developed extend from the main haulage horizon to the return airway horizon if possible, and staple shafts are therefore avoided or reduced to one per district. If there is no haulage installation in the return airway horizon, one staple shaft is advisable for the transport of men, machinery and stowage material. In this case, materials and supplies are transported on the main haulage level to the staple shaft bottom and via the staple shaft to the return airway horizon, where they can be distributed as required.

Fig. 57 is a cross-section through a district in which there are six seams dipping at 35 degrees. The vertical distance between horizons is 150 yards, the length of the faces is 300 yards, and the angle of dip on the face is 28 degrees; thus the faces have a diagonal layout. Each face is equipped with retarding disc conveyors. Generally, diverging faces on the strike are worked simultaneously in each seam in descending order. The total working length in one direction of strike is 900 yards, and in the opposite direction 1,100 yards. The daily output per face ranges between 160 and 300 tons according to the working conditions and the thickness of the seam, which varies from 2 to 3 feet. Since 1,000 tons per day can be considered a reasonable dis-

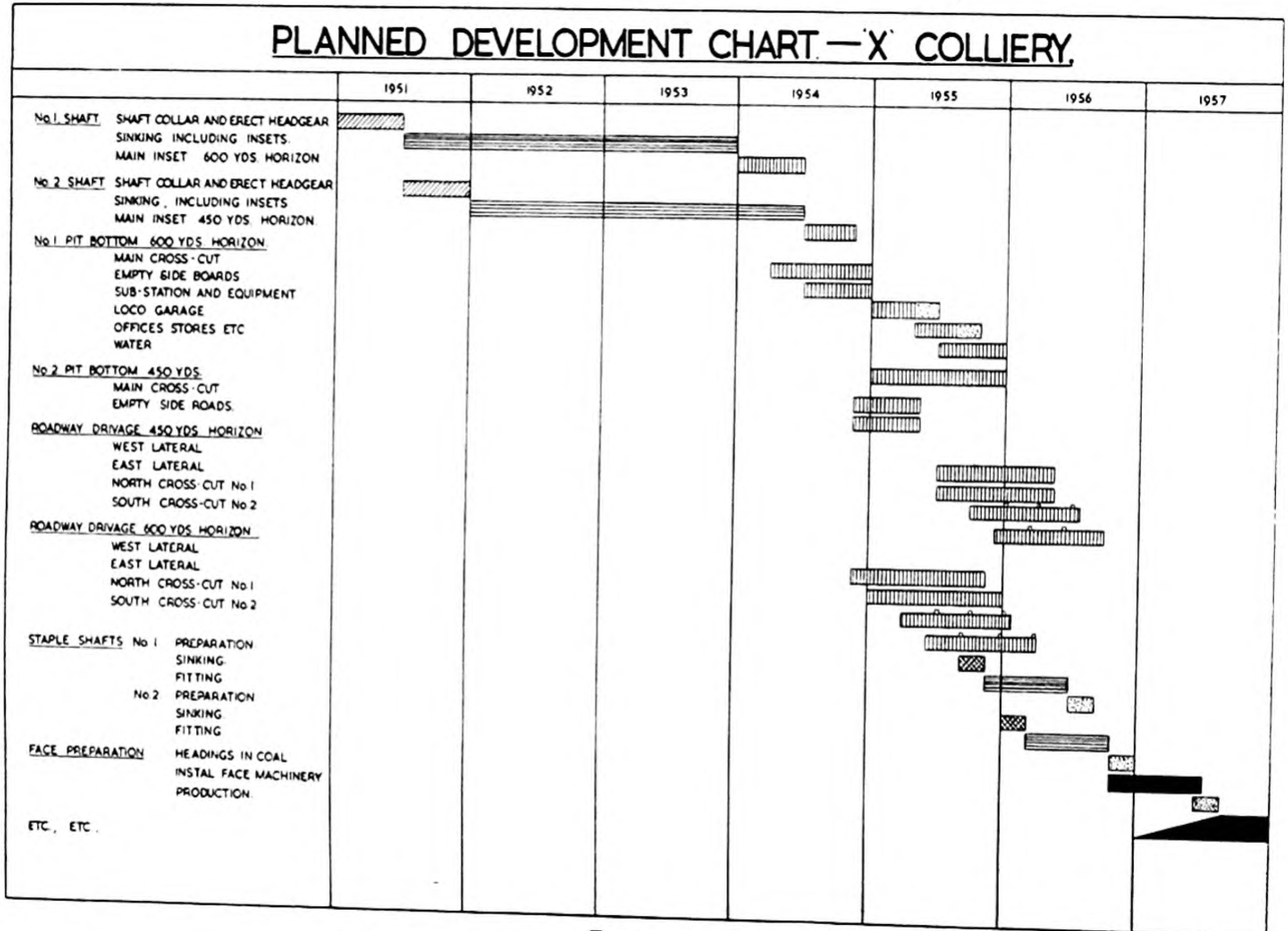

PLANNED DEVELOPMENT CHART—'X' COLLIERY.
1951
1952
1953
1954
1955
1956
1957
No. 1 SHAFT
SHAFT COLLAR AND ERECT HEADGEAR
SINKING INCLUDING INSETS.
MAIN INSET 600 YDS. HORIZON
No. 2 SHAFT
SHAFT COLLAR AND ERECT HEADGEAR
SINKING, INCLUDING INSETS
MAIN INSET 450 YDS. HORIZON
No. 1 PIT BOTTOM 600 YDS HORIZON
MAIN CROSS-CUT
EMPTY SIDE BOARDS
SUB-STATION AND EQUIPMENT
LOCO GARAGE
OFFICES STORES ETC
WATER
No. 2 PIT BOTTOM 450 YDS.
MAIN CROSS-CUT
EMPTY SIDE ROADS.
ROADWAY DRIVAGE 450 YDS HORIZON
WEST LATERAL
EAST LATERAL
NORTH CROSS-CUT No. 1
SOUTH CROSS-CUT No. 2
ROADWAY DRIVAGE 600 YDS HORIZON
WEST LATERAL
EAST LATERAL
NORTH CROSS-CUT No. 1
SOUTH CROSS-CUT No. 2
STAPLE SHAFTS No. 1
PREPARATION
SINKING.
FITTING
No. 2
PREPARATION
SINKING.
FITTING
FACE PREPARATION
HEADINGS IN COAL
INSTAL FACE MACHINERY
PRODUCTION.
ETC., ETC.

FIG. 55

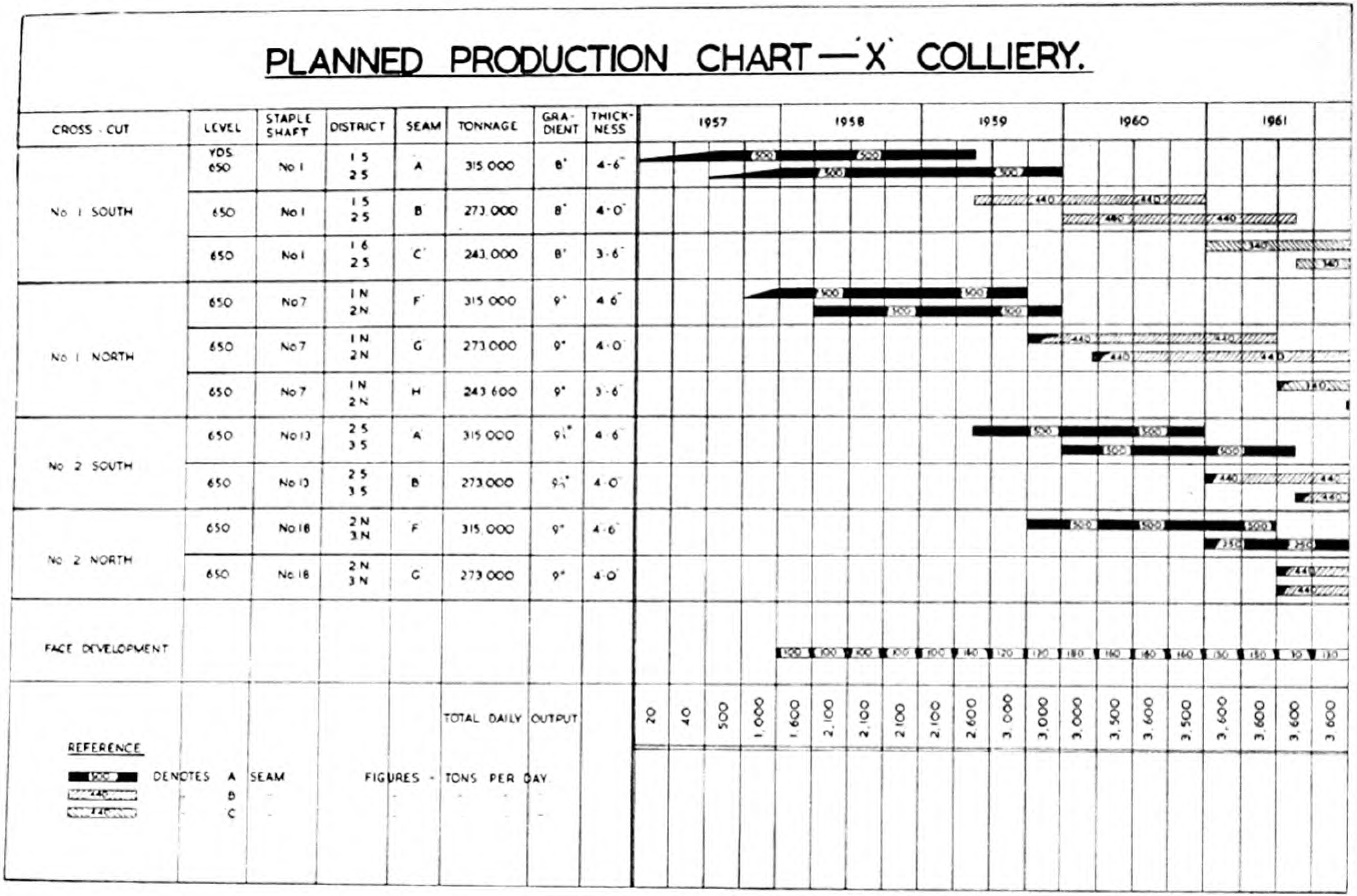

PLANNED PRODUCTION CHART—'X' COLLIERY.
CROSS - CUT
LEVEL
STAPLE SHAFT
DISTRICT
SEAM
TONNAGE
GRA-DIENT
THICK-NESS
1957
1958
1959
1960
1961
No 1 SOUTH
YDS. 650
No 1
1 S 2 S
A
315,000
8°
4-6"
650
No 1
1 S 2 S
B
273,000
8°
4-0"
650
No 1
1 6 2 S
C
243,000
8°
3-6"
No 1 NORTH
650
No 7
1 N 2 N
F
315,000
9°
4-6"
650
No 7
1 N 2 N
G
273,000
9°
4-0"
650
No 7
1 N 2 N
H
243,600
9°
3-6"
No 2 SOUTH
650
No 13
2 S 3 S
A
315,000
9½°
4-6"
650
No 13
2 S 3 S
B
273,000
9½°
4-0"
No 2 NORTH
650
No 18
2 N 3 N
F
315,000
9°
4-6"
650
No 18
2 N 3 N
G
273,000
9°
4-0"
FACE DEVELOPMENT
TOTAL DAILY OUTPUT
20
40
500
1,000
1,600
2,100
2,100
2,100
2,100
2,600
3,000
3,000
3,000
3,500
3,600
3,500
3,600
3,600
3,600
3,600
REFERENCE
DENOTES A SEAM
B
C
FIGURES - TONS PER DAY

FIG. 56

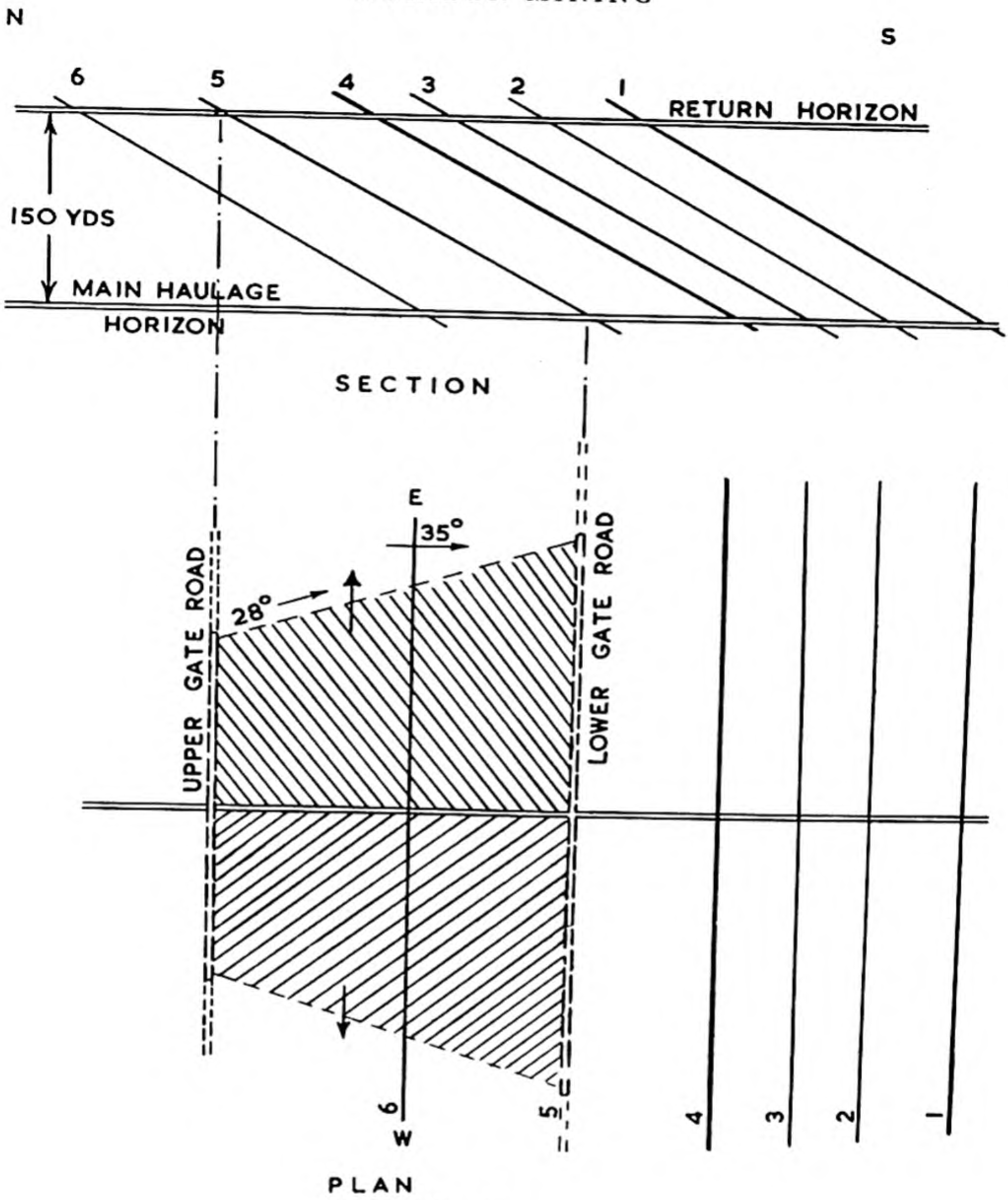


FIG. 57

trict output, 3 or 4 faces must be worked simultaneously in two adjacent seams. It must be considered, however, that the simultaneous working of adjacent seams will affect the conditions in the lower and upper seams, and working of the lower seam cannot be started until some months after commencing extraction in the upper seam. In most cases a period of a few months can be considered sufficiently

long to allow the strata to regain their stability. Under these conditions, any assistance to be gained in extracting the lower seam by virtue of an effective pressure distribution is regained, and the extraction of the lower seam under the upper seam waste does not affect the working of the upper seam. The production chart, Fig. 58, shows the sequence of working in the various seams and the period necessary for systematic face development. In this case the developments required in rock are not considered and, as can be seen from the chart, the district produces an output of about 1,000 tons per day over a period of approximately seven years.

(*h*) *Layout of faces in steep measures.* In steeply inclined seams, where the dip is greater than 1 in 1·2, as distinct from semi-steep seams, it is not possible to develop faces extending between horizons. The length of such faces would be too great or the horizon interval severely restricted. The vertical distance between the horizons must therefore be sub-divided into 2, 3 or 4 sections, each 60 to 90 yards in depth and separated by sub-levels (ref. Chapter 1, Part II, Section 4 (*c*)). Each of these sections will incorporate two faces laid out diagonally and between 100 and 200 yards long, according to the dip.

There are three different methods of working these vertical sections in steeply inclined seams, which are:

Method 1. This is illustrated in Fig. 59, and entails working the sections between levels in ascending order, each section consisting of two diverging faces won out over the length of the panel from the sub-level cross-measure drifts. Each section is worked independently. This method has the advantage that the development required before the faces commence takes the least possible time, and the gate roads which are driven for waste transport in the first instance are later used for the transport of coal from the next and higher face between sub-levels. The disadvantages outweigh the advantages, as there is often excessive gas emission from the upper sub-level gate road which is driven in the solid. The last remaining coal pillar lying at the extremities of the upper section will probably be surrounded completely by waste, as the coal has been extracted above the upper airway horizon, below the sub-level horizon and probably on one or both sides of the panel. The concentration of pressure on these pillars causes roof trouble and possible 'rock bursts' and so makes the extraction of the pillars difficult. This method cannot therefore be generally recommended.

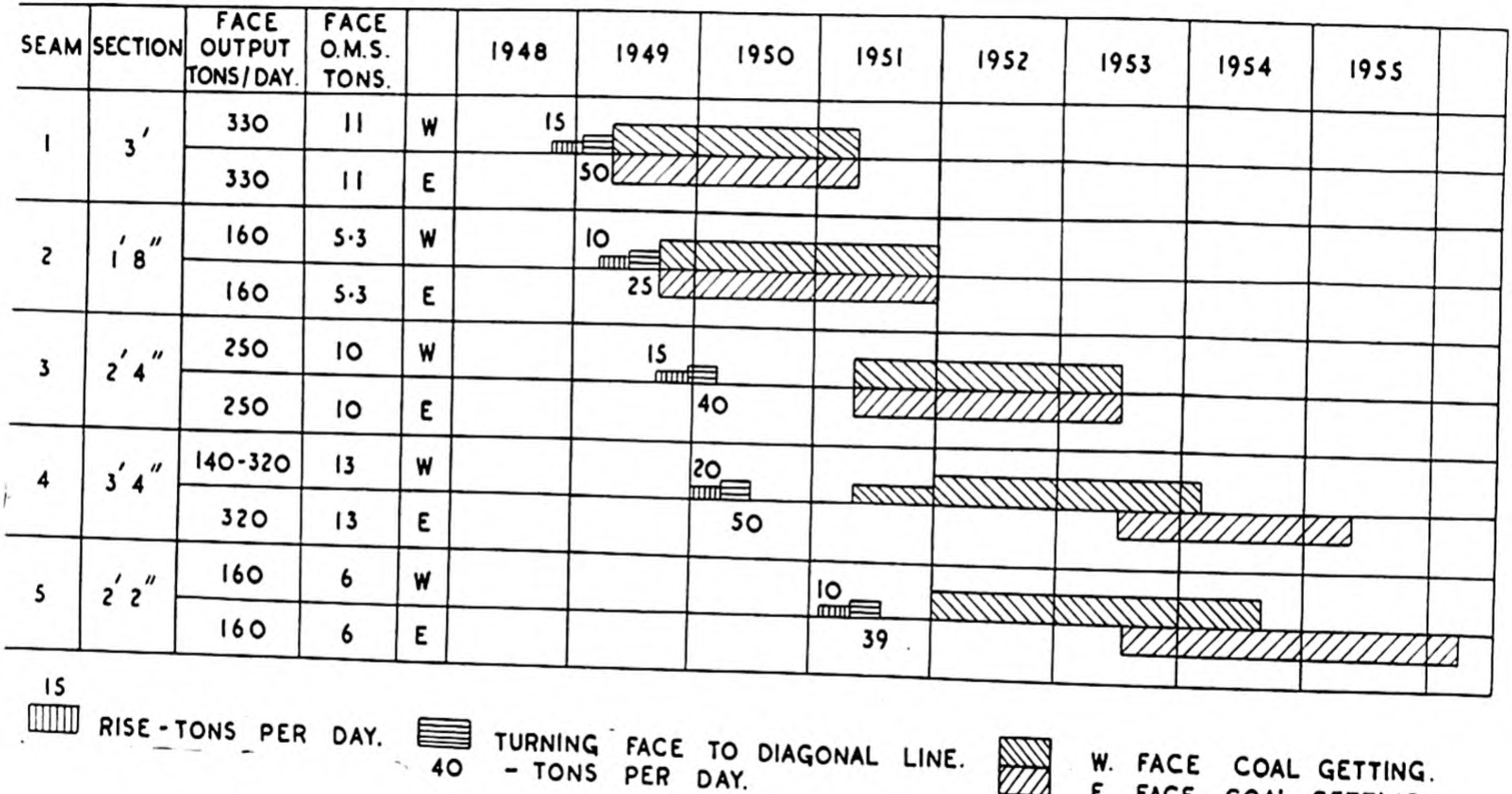

SEAM
SECTION
FACE OUTPUT TONS/DAY.
FACE O.M.S. TONS.
1948
1949
1950
1951
1952
1953
1954
1955
1
3′
330
11
W
15
330
11
E
50
2
1′ 8″
160
5·3
W
10
160
5·3
E
25
3
2′ 4″
250
10
W
15
250
10
E
40
4
3′ 4″
140-320
13
W
20
320
13
E
50
5
2′ 2″
160
6
W
10
160
6
E
39
15
RISE - TONS PER DAY.
TURNING FACE TO DIAGONAL LINE.
40 - TONS PER DAY.
W. FACE COAL GETTING.
E. FACE COAL GETTING.

Fig. 58

Method 2. This reverses the sequence of extraction from the upper section in descending order. The disadvantages previously discussed do not generally arise, but other difficulties are apparent. The gate roads which are first used for coal transport can be maintained only with great difficulty and, in most cases, are not in a suitable condition for further use as waste transport roads for the next two lower faces.

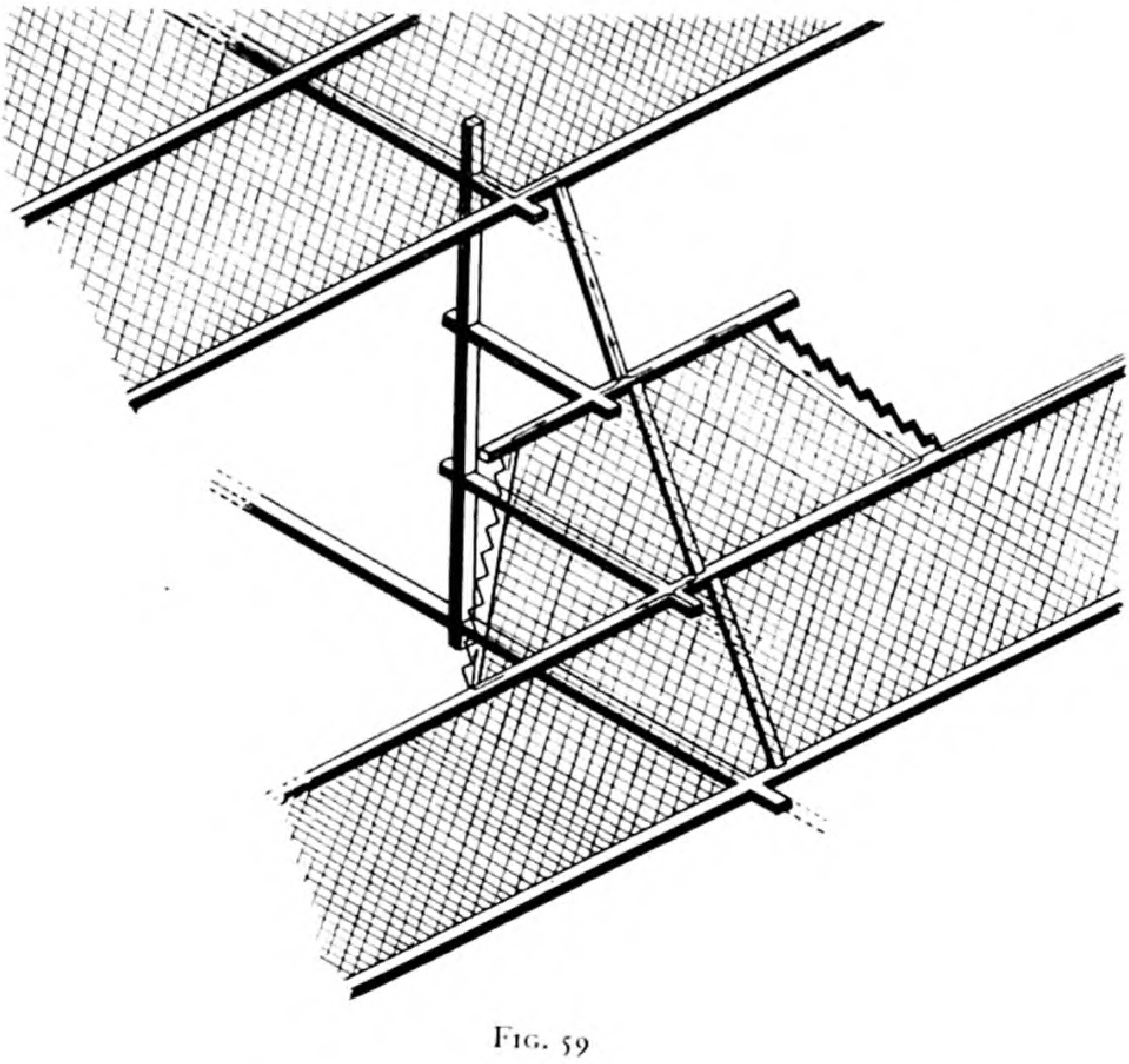


FIG. 59

New gate roads must therefore be driven through the seam below the upper goaf. This second method is therefore only practicable if the seam is thick.

Method 3. This method may be sub-divided into two systems in which all sections are worked simultaneously as shown in Figs. 60A and B.

The method shown in Fig. 60A, in which extraction is carried out in descending order, has a major disadvantage. Coal crush occurs at

the corners formed between the coal face and the upper gate roads, and this coal may run down on to the working face. Ventilation is also difficult at these junctions, where pressure losses are high, and leakage through the waste to the upper return airway may take place.

The alternative method shown in Fig. 60B is considered to be the

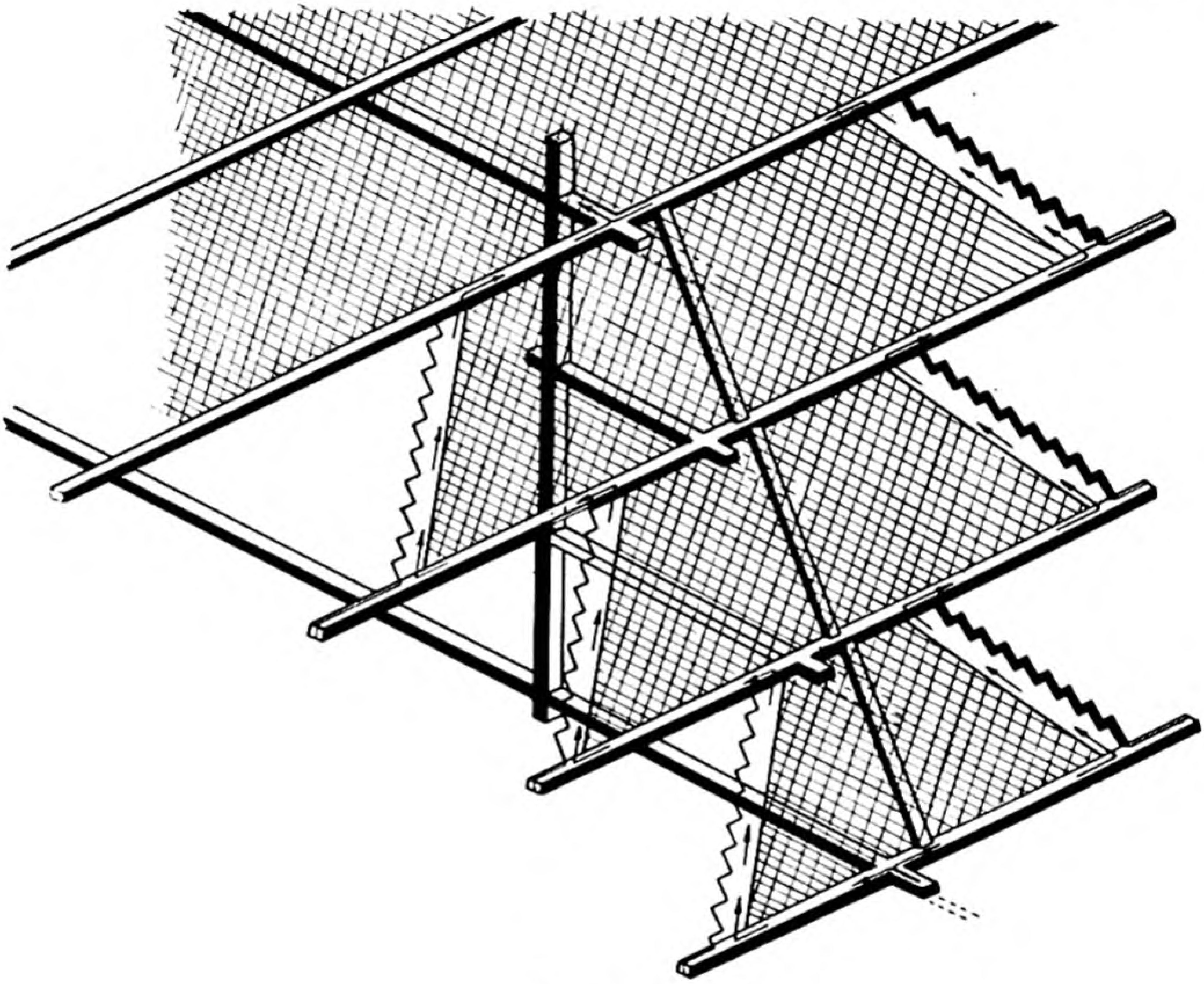


FIG. 60A

best. The sharp junctions are avoided and the face line is almost continuous between horizons. Rock bursts are infrequent, the rock pressure not being concentrated on the remaining coal pillars since the roof on the whole unit has not yet settled as in the case described in Method 1.

As in flat and semi-steep seams, a high output must be projected and it may range from 700 to 1,200 tons per day, according to the thickness and number of seams to be worked and other mining conditions, such as the transport of stowage material, ventilation and

the winding capacity of the staple shafts. If the vertical interval between horizons is sub-divided into three sections by two sub-levels, three pairs of diverging faces, or six faces in all, may be worked simultaneously and an output of more than 700 tons per day can easily be achieved. It is also possible to work two adjacent seams at

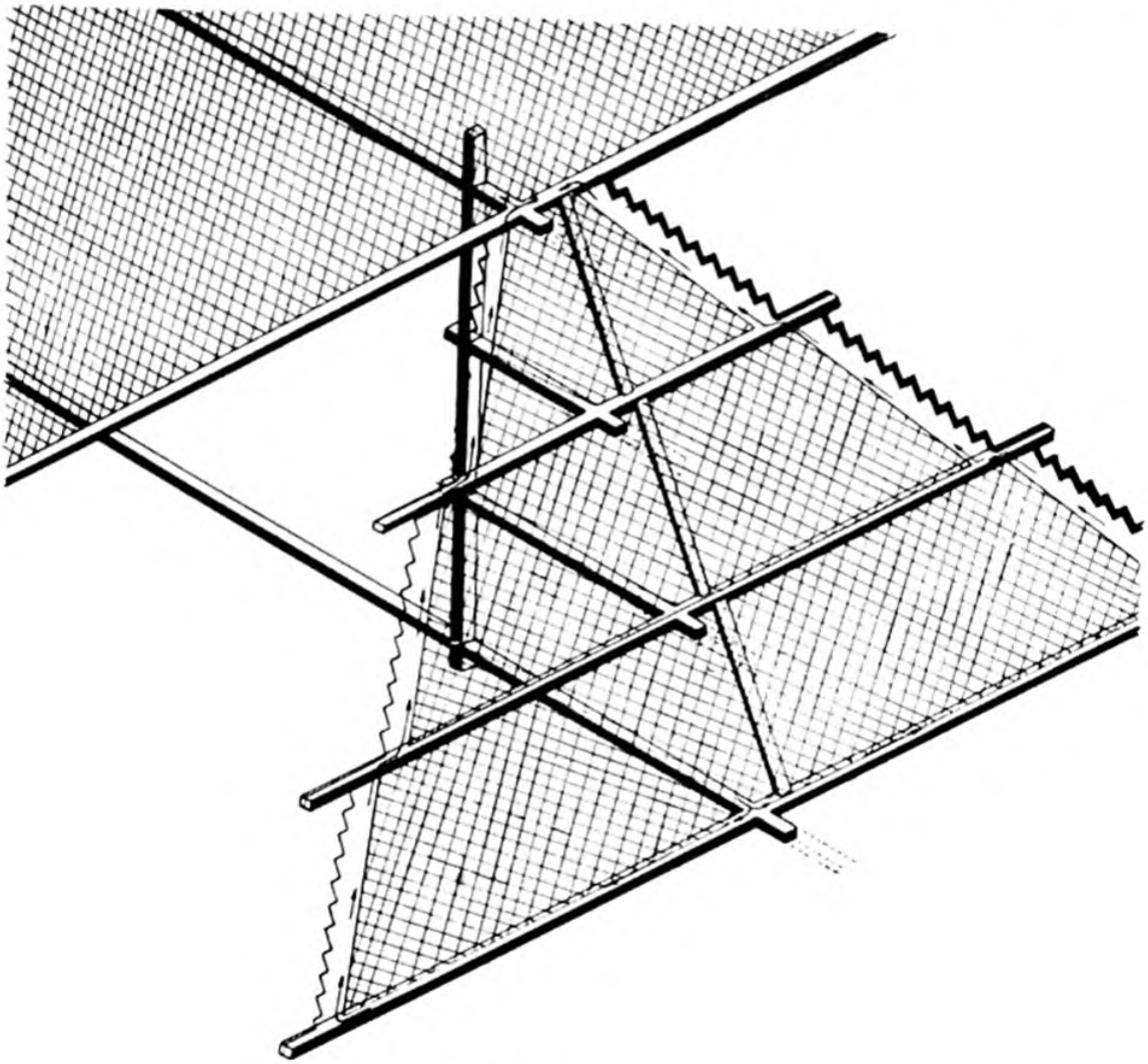


FIG. 60B

the same time if, before the second seam is worked, several months are allowed to elapse after extraction has commenced in the first seam.

Section 7. Sequence of Working the Seams

The sequence of working is dependent upon several factors, of which the extent of coal reserves and the mutual influence of working in adjacent seams are the most important.

In the conventional dip-mining system practised in Great Britain and the U.S.A., the practice has invariably been to work the best

seams first, both from the mining and marketing points of view. This has resulted, in many cases, in a seam or seams remaining virgin above or below worked-out areas in other seams. These seams have then been developed at a later stage, depending upon the method of working adopted and the distance between the seams. This is especially the case if longwall has been applied in a lower seam, and if the distance between the upper immediate seam and the lower goafed area is in the region of from 20 to 30 yards. Where the room-and-pillar, or bord-and-pillar, system of mining has been employed, this method of working has often had a detrimental effect on the later development of an upper seam, especially in the case where the seams are close together, due to the irregular pressure distribution over the coal pillars formed.

With the horizon mining system, any seam or section of seam which is not worked during the same period as extraction has been carried out in adjacent seams must, in most cases, be considered as lost. This is definitely the case where staple shafts and main roads giving access to the remaining unworked areas are abandoned. It is a golden rule in the horizon mining system that, in order to exploit the reserves to the best advantage, the sequence of seam working should be from top to bottom or in descending order.

Ordered sequence of working from the upper seams downwards has another advantage in that working costs and profits are evenly distributed over the period of life of the reserves. If only the best seams are worked first, costs are low and profits high, but this is inevitably followed by a period in which the working costs are higher and the resulting profits reduced. In many cases this results in poorer seams being considered uneconomic to work. A reasonable balance should be planned between costs and profits, and full advantage taken of the available reserves, giving a longer life to the undertaking and conserving resources.

This principle is not to be understood to imply that in every case an upper seam must be completely worked out before a lower seam can be developed, or that a lower seam cannot be developed before an upper seam.

It is generally the case that inter-action between the extraction in adjacent seams is of greater importance, and that every effort should be made to make full use of the natural forces in the strata in order to assist working in each seam. In other words, if a lower seam is worked

simultaneously with an upper seam in the same district, then a more or less continuous state of disturbance occurs, in which difficulty may be experienced in working the coal in the upper seam, which is 'winded', and deterioration of roof and floor conditions also takes place. Likewise, working in the lower seam is affected by the extraction of the upper seam, but not to the same extent.

Similar effects may be experienced if the period between working the seams is too short. Settlement of the strata is not given time to take place before the next seam is worked, that is, before the wastes have had time to consolidate and are capable of taking their full steady load again. When the seam being worked is thick and the roof strong, this stage takes longer to accomplish than when a thin seam

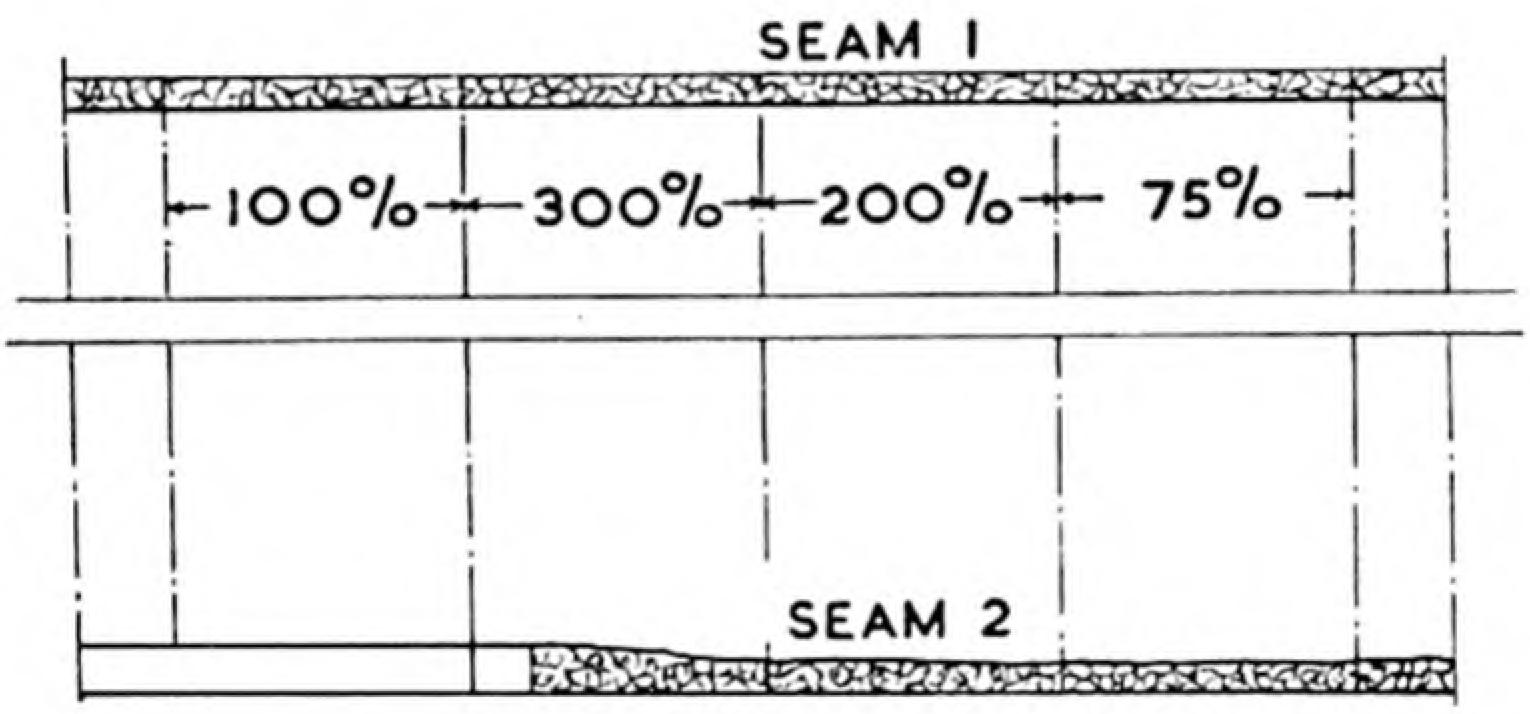


FIG. 61

with a weaker roof is worked. The extent or area of working is also important, especially when considered with rate of advance and extraction. Where extraction is slow, strata settlement takes place slowly and a longer period must be allowed before the upper seam can be worked satisfactorily. A systematic and steady rate of advance is important from every point of view. Fig. 61 illustrates the extent to which previous working in a lower seam may influence output per manshift and piecework rates in the subsequent working of an upper seam. The piece-work rates are lowest above the unworked area and above that section of the waste in the lower seam which has fully consolidated. The rates are highest where working has taken place recently and where the waste has just been laid down. In some instances, however, it may be desirable to work a lower seam first, so as to prevent the occurrence of 'pressure bursts' in the upper seam. This order of working may also be carried out where a bigger size

of coal is required from the upper seam. The lower seam is worked first, followed by extraction in the upper seam, in which case a reduced effective pressure is applied to the upper seam.

The quantity of methane to be adequately diluted is also of extreme importance, especially when the upper seam is thinner than the lower seam. In these circumstances, due to the smaller sectional area of the faces in the upper seam, it is more difficult to ventilate the faces efficiently. The thicker, lower seam should be worked first, thereby reducing the probable quantity of gas to be dealt with during the later working of the thinner, upper seam.

Section 8. *Consideration of the Size of a Cross-measure Drift Unit*

The establishment of a minimum working cost is a major consideration in the distribution of the production faces within the colliery, and therefore affects the output capacity of the cross-measure drift unit. In order to achieve a minimum working cost, it is imperative that the best possible use be made of the grid roads developed and the equipment installed. A high rate of utilisation can be achieved only if the roads, staple shafts and machinery installed are worked to their capacity and the unit output is the best that can be obtained. Roads and machinery have a fixed depreciation which is almost independent of the output; if the output is small, this sum must be charged against a small tonnage and the overall cost per ton is high. Conversely, the cost per ton is reduced with increased output.

It may be construed from these factors that high concentration to a limited part of the workable reserves is desirable, even in a single cross-measure drift unit, in order to reduce the total length of roads to be maintained and to maintain a high rate of utilisation.

One disadvantage of high concentration is the mutual influence of coal faces in adjacent seams. The effects of strata pressure near staple shafts and on main roads must be considered. The increase in the maintenance cost resulting from an excessive number of faces per district has to be taken into consideration. It may also be impossible to provide an adequate quantity of air for face ventilation if the number of faces is too high, without resorting to an uncomfortably high air velocity under the normal restricted cross-sectional area conditions at the face. Consequently, ventilation conditions will set

a limit in most cases to the number of faces which may be worked conveniently within a cross-measure drift unit.

It is therefore necessary to distribute the coal faces in such a manner as to minimise their effect on neighbouring roads and on each other, and to ensure good ventilation standards. In order to reach the highest rate of utilisation, the faces required should be distributed between neighbouring cross-measure drifts so as to increase the output per cross-measure drift unit to the optimum extent depending upon these considerations. The output is mainly limited by two factors, the more important of which is the quantity of air which can be passed through the cross-measure drift, and this depends on the cross-sectional area of the road and the velocity of the air current. The roadway section in many cases is limited to 100–130 square feet and the air velocity to 500–700 feet per minute, giving a quantity of approximately 50,000–90,000 c.f.m., which is normally sufficient to meet the requirements of an output of from 1,000 to 1,500 tons per day. In cases where the gas emission is high and irregular geological conditions occur, this level of production is very often reduced.

Section 9. Consideration of the Size of the Colliery Unit

(*a*) *General remarks.* The 'size' of a coal mine is dependent upon the output per day or per annum, and the choice of the correct size for the unit is essential for the economic operation of the mine. It depends upon a variety of factors, the first consideration being the initial capital investment required to develop the mine. This, in turn, is governed by the number, quality and thickness of the seams available, their depth and the general geological formation, and the nature of the overburden where special conditions require costly methods of sinking.

The seams in the British coal fields are predominantly in flat formations, and in many cases no overburden exists which calls for special methods of shaft sinking. Many of the fields produce coal from shallow depths and, in many cases, the coal quality is such that expensive coal-cleaning equipment is not required. Because of these advantages, many small mines having a daily output of from 500 to 700 tons per day have been developed, and the tendency has been to produce from smaller units than is usually the case in Continental practice.

This small unit is now being deposed in favour of a larger capacity unit and the general trend is towards a mine having a daily output of from 3,000 to 6,000 tons. It is anticipated that this development will continue.

In the U.S.A. the majority of mines are small units, each producing about 300 tons per day, and the number of mines producing a few thousand tons per day is also increasing, with some mines producing as much as 10,000 to 15,000 tons per day. The geological conditions in the U.S.A., while being similar to those in Britain, are more favourable with regard to depth and average seam thickness. The average seam thickness can be taken as 6 to 7 feet in the U.S.A., as 4 feet in England and as 3 feet in Scotland.

This development to the larger mine in Britain and the U.S.A. is due mainly to the recognised fact that a large mine can be run more economically than a number of smaller mines producing the same total output. Many improvements in mining technique, especially in regard to underground haulage, have taken place in recent years.

On the Continent, larger producing units were introduced a comparatively long time ago. In the Ruhr district the average daily production from colliery units in 1950 was 2,500 tons, while only a few mines produced less than 1,000 tons per day. About fifty mines have daily outputs of more than 3,000 tons per day, of which eight mines have an output between 5,000 and 6,000 tons per day, and two mines more than 7,000 tons per day, three mines 8,000 to 9,000 tons per day, and one mine (Zollverein) 12,000 tons per day. Several collieries having a daily output of 10,000 tons are in course of construction.

In Holland, where similar conditions exist as in the Ruhr district of Germany, twelve mines produce about 13 million tons per annum, givingan average daily output of 3,800 tons from each mine. Thefour Dutch state mines, which are the most recently developed of the Dutch collieries, have an average daily output of 6,000 tons. Maurits mine is the largest single production unit, with an average output of from 9,000 to 10,000 tons per day. In the Belgian coal fields there is a marked difference between the older mines in Southern Belgium and the new mines in the Campine. The mines in Southern Belgiumeach produce about 1,000 tons per day, whereas the average daily output of the Campine mines is more than 4,000 tons per unit.

There are several reasons why Continental mines accepted the prin-

ciple of the larger production unit at an early date. The seams are not so easily accessible as in Great Britain and the U.S.A., and the reserves extend in depth, rather than laterally over wide areas. The seam density is comparatively high. The carboniferous measures are, as a rule, covered by an uncomfortable recent strata with quicksand as an overburden, so that sinking is both deep and expensive. The seams are of lower quality than in Great Britain and coal-cleaning plants are a necessity at every mine. The capital expenditure required for and the operating costs of coal-cleaning plant for one large mine are less than for a number of small mines, similarly equipped, to deal with the same total capacity. The horizon mining system, which is universally adopted, implies a certain minimum daily production, to cover the initial capital outlay and working costs for roads and staple shafts, in order to keep the cost per ton at a reasonable level. If the seams are not horizontal, only a horizon mining system, with its level roads in the strata, can provide the means of utilising the efficient and cheap system of locomotive haulage.

It may be said that, if production in Great Britain is to be maintained at, say, 240 million tons per annum, this output could be obtained from a reorganisation programme, the aim of which would be to have 240 mines each producing an average annual output of 1 million tons, or an average daily output of 4,000 tons.

(*b*) *The influence of coal reserves on the size of the area and on the output of a colliery unit.* The coal reserves are usually given in tons, assuming a limit to the workable depth of 4,000 feet, and a minimum working thickness of 2 feet or 1 foot 6 inches. The reserves can be computed on a basis of tons per foot of total seam thickness per acre. A usual figure, taking into consideration the presence of faults and loss in working, is 1,350 tons per foot per acre, or it may be expressed as 120 tons per inch per acre. Such expressions can only relate to the actual total thickness of coal existing within a field, and have no direct correlation with depth. If the reserves extend over a depth of D ft., and the total seam thickness is T ft., the reserves have a density of $\frac{T}{D} \times 100$ per cent. In Great Britain the average 'seam density' ranges between 0·5 and 2 per cent.

In comparison, the average seam density in the Ruhr district of Germany is about 1·8 per cent., ranging between 0·8 and 4 per cent.

in certain cases. On the Continent, the reserves are usually expressed in tons per sq. km. For purposes of illustration, these values have been converted to tons per acre as the British equivalent. For example, if the coal reserves down to a depth of 1,200 yards are assumed to be 60,000 tons per acre, an area of 500 acres would produce a reserve of 30 million tons; such an area is too small for an economic production unit, since the capital and deprecation costs per ton of saleable output would be too high (ref. Part II, Section 9 (*e*)). A modern mine is expected to have an anticipated life of about one hundred years. If a daily production of 4,000 tons is planned, giving an annual output over 250 working days of 1 million tons, an area of reserves of 100 million tons, at least, is required. This will need a reserve area of 1,600 acres at a figure of 60,000 tons per acre.

Area of Colliery 'Take' Acres	*Coal Reserves Million Tons*		*Life in Years at Different Outputs in Tons per annum and per day*							
			0·5 million Tons p.a. 2,000 Tons p.d.		*0·75 million Tons p.a. 3,000 Tons p.d.*		*1 million tons p.a. 4,000 Tons p.d.*		*1·5 million Tons p.a. 6,000 Tons p.d.*	
	Of Colliery	*Between Two Horizons*	*Of Colliery*	*Main Haulage Horizon*	*Of Colliery*	*Main Haulage Horizon*	*Of Colliery*	*Main Haulage Horizon*	*Of Colliery*	*Main Haulage Horizon*
500	30	5	60	10	40	6·6	30	5	20	3·3
1,000	60	10	120	20	80	13·2	60	10	40	6·6
1,500	90	15	180	30	120	20·0	90	15	60	10·0
2,000	120	20	—	—	160	26·6	120	20	80	13·3
2,500	150	25	—	—	—	—	150	25	100	16·6
3,000	180	30	—	—	—	—	180	30	120	20·0

It is evident from the above table, based on the same figure of reserves, that a 500-acre 'take' or concession is required for a colliery life of sixty years at a daily output of 2,000 tons, 750 acres for 3,000 tons per day and 1,500 acres for 6,000 tons per day. The larger the

planned output, with a given specific value of reserves per acre, the bigger the mining 'take' has to be, or the life of the colliery will be reduced. For a life of ninety years at 4,000 tons per day, 1,500 acres are required, and 2,250 acres for an output of 6,000 tons per day. From the same table, conclusions may be drawn regarding the life of the levels. The depth of 1,200 yards has been assumed which, divided into six main haulage levels, represents a horizon interval of 200 yards. Above every haulage level two or three seams are assumed to be present, having a total thickness of 8 feet, which gives a seam density of 8/600, or 1·3 per cent. The table illustrates that, with areas of 500 acres and 1,000 acres, at a daily production of 4,000 tons, the life of a level is only five and ten years respectively, which periods are too short. A life of from fifteen to twenty years is required in order to cover the depreciation cost per ton of output and the driving cost incurred in making the level. With a life of fifteen years per level, 750, 1,500 or 2,250 acres are necessary to cover daily outputs of 2,000, 4,000 and 6,000 tons respectively. The same figures for a life of twenty years per level would be 1,000, 2,000 and 3,000 acres.

(*c*) *The rate of area utilisation.* It is quite obvious that the whole of the concession or 'take' need not be worked in order to attain the planned output. It is not advisable, on the grounds of both ventilation and haulage, to spread the extraction over too wide an area (ref. Chapter 1, Part II, Section 5). The part of the concession which is actually mined may be called the 'working area'. For the proper development of the mine, the optimum output must be produced from that working area. Assuming a rectangular area of 4,320 × 2,185 yards, i.e. 2,000 acres, divided into ten working panels on the strike, each with a working length of about 220 yards, every panel has a working area of 200 acres, *vide* Fig. 62.

The concession is to be worked by three cross-cuts, so that there will be six working districts, each having a striking length of 720 yards. By developing double-unit faces having 220 yards on each side, or 440 yards long, in a seam four feet thick and having a daily advance of five feet, the daily output per double unit would be 720 tons of saleable coal. For a daily production of about 4,000 tons, six double units would be required, and these would be developed as diverging faces, two from each cross-cut, as illustrated in Fig. 62.

The extraction rate per acre of concession would be 4,000/2,000,

or 2 tons per day per acre. This figure varies between 0·5 and 6 tons per day per acre in the Ruhr district, so that the value of 2 tons per day per acre is comparatively low. It is possible, in the example being discussed, to increase the output from the colliery to 5,000 or 6,000 tons per day. Double-unit faces could be won out from the cross-cuts in the southern extent of the working area, with a daily advance sufficient to increase the extraction rate per acre. Three double units could be developed which would increase the daily production, at a five feet daily advance, by 2,000 tons and thus give a total daily production of 6,000 tons from the colliery. The extraction rate

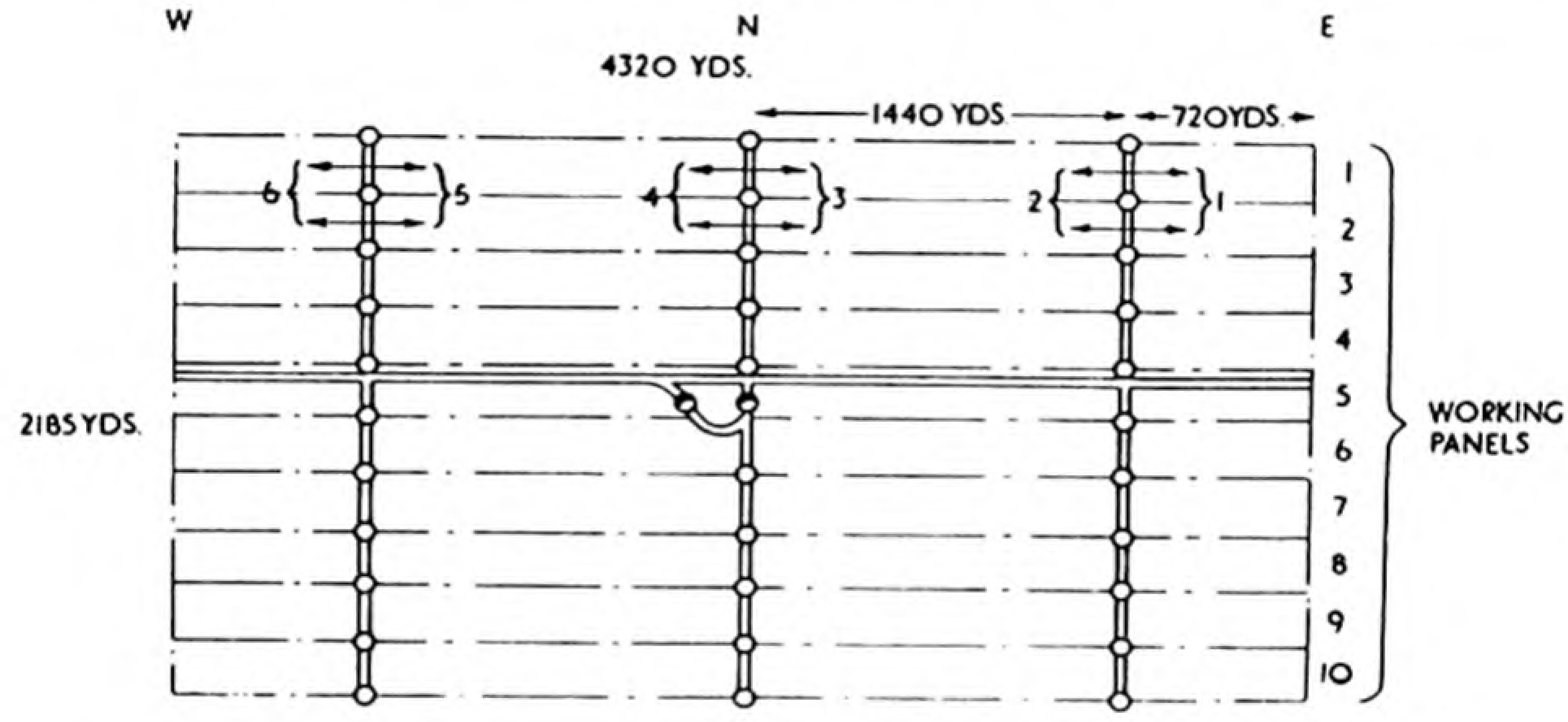


FIG. 62

per day per acre would then be 6,000/2,000, or 3 tons. The same objective could be achieved by working six single-unit faces, each about 200 yards long. A combination of the previous methods could also be used to obtain the same objective.

The working area is naturally only part of the mining concession. In the example quoted, the gradual development of double-unit faces from the shaft district will begin production, with subsequent faces coming into operation until the full production rate is achieved. The life of a double-unit face in this instance is 440 days, so that several of these faces will be worked out when other districts begin extraction.

The new faces have to be developed according to a production schedule so as to come into production as other units are worked out. Rises are developed in the second or third seam between the levels, cross-cuts are extended and staple shafts sunk, in order to maintain development in advance and guarantee a regular production rate.

(*d*) *The influence of area and output on capital expenditure.* The capital expenditure required for a colliery unit rises with increasing output and the area of the concession to be worked. Two shafts are necessary for any mine, while three, four, five or six shafts may be required for a larger mine, especially in the case of a combined mine, if only for ventilation purposes. The total length of underground roadway drivage is increased and the machinery units on the surface and underground have to be larger, more efficient and more extensive.

In order to investigate this position, three 'takes' are considered of 1,500, 2,000 and 3,000 acres, with assumed daily productions of 3,000, 4,000 and 6,000 tons respectively. The area effect or extraction rate per day per acre is the same in all three cases. The geological conditions, thickness of seams and coal reserves are considered to be identical. The necessary capital expenditure for the purchase of the coal reserves and the surface area required, together with the cost of main shaft and staple shaft sinking, drivage of roads, cost of machinery and equipment needed, both on the surface and underground, will now be considered.

The purchase cost of surface requirements does not increase at the same rate as the output, but at a lower rate. The surface plant for a 4,000 tons per day mine would require, say 15 per cent. more area than for a mine with an output of 3,000 tons per day, while to deal with an output of 6,000 tons per day, an area, say, 40 per cent. greater would be required.

When considering shaft-sinking costs, the main shafts will require to be of greater diameter than the secondary shafts of a combined mine when dealing with the larger 'take' of a high-production unit. These secondary shafts will be sunk at a lower cost than the main shafts, and shaft-sinking costs will, as a rule, not increase at the same rate as the output. The costs per annual ton of output decrease with increased output, and the decrease will be the more pronounced the greater the difference between the cost of main shaft and secondary shaft sinking. This is the case in the Ruhr district, where special methods of shaft sinking such as freezing and cementation systems have to be employed.

The expensive pit bottom is relatively cheaper for a large output than for a smaller one.

The total length of underground drivage, both of roadways and staple shafts, increases with increase in output, and consequently this

cost will increase at the same rate as the output. Thus, on the whole, capital expenditure per ton of annual output on the development of the underground grid of roadways and staple shafts can be assumed to decrease slightly.

Some distinction must be made in the consideration of machinery and equipment requirements between face-machinery, haulage equipment and ancilliary items, such as compressed-air mains, electric cables, etc. The number of face units and their necessary equipment will increase with the output, so that the cost per annual ton will remain more or less constant. The capital outlay on pumps, pipe-lines, etc., will increase with output but at a reduced rate. The capital expenditure required on machinery will therefore decrease slightly with increase in output per annum.

When considering surface plant installations for the mine, there is a greater difference in the requirements with increase in output. The cost of winding engines and equipment does not increase at the same rate as the output but, for a 6,000 tons per day mine, may require an expenditure of about 40 per cent. more than a mine having an output of 3,000 tons per day. Coal-cleaning plant requirements can be estimated to be about 60 per cent. greater for the same output difference of from 3,000 to 6,000 tons per day. In the case of other surface plant, comparative estimates would be: for ventilation, 25 per cent.; boiler plants, engine house and switch gear, 40 per cent.; loading arrangements and surface lines and sidings, 50 per cent. The cost of surface installations at secondary shaft sites is somewhat different. The equipment cost of a number of secondary shafts increases at a greater rate than the output. Thus, the capital cost of surface plant at secondary shafts may be from two and a half to three times more for a mine having a daily output of 6,000 tons than for a mine having a daily output of 3,000 tons. The cost for surface plant per ton of annual output at 6,000 tons per day mine may be approximately 60 per cent. more than the cost at a mine producing 3,000 tons per day.

The capital cost of underground and surface plants at mines of various outputs therefore increases with output but not at the same rate. A 6,000 tons per day mine will have a capital cost which is 60–70 per cent. higher than that for a 3,000 tons per day unit. If the main shaft-sinking costs are high in both cases—that is, if the freezing method is employed—the relative cost difference is reduced to 50–60 per cent.

(e) *Capital costs and expenditure*. Very often the capital expenditure required to develop a new mine and the investment, or capital, costs are taken as being the same, but this is not the case. A new mine is unlike a factory which, when construction is completed, can come into full production almost immediately. The development to full production of a new colliery may take eight, ten or fifteen years. Production increases progressively up to the planned output as the main

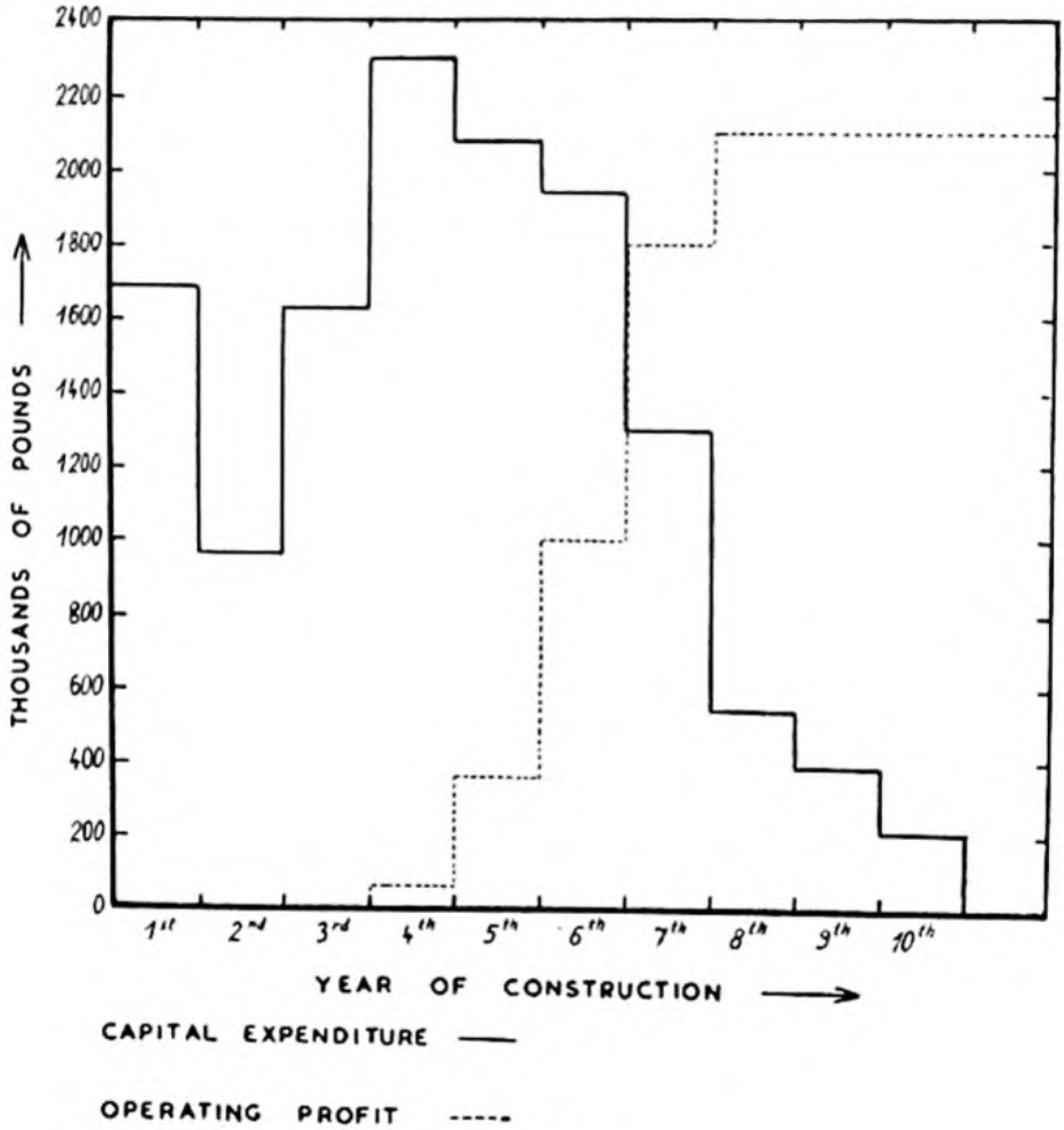


FIG. 63

development proceeds. Thus, even while undergoing development, a mine has a certain revenue from the output being produced. These cash returns do not diminish the investment, or capital, cost but do reduce capital requirements. Interest has to be paid on the capital spent during the period of development, and this interest may increase the capital expenditure considerably.

The exact influence of these factors on capital requirements has to be analysed by drawing up a progressive time and finance schedule in

every case. The diagram in Fig. 63 shows such a schedule for a 6,000 tons per day mine, and the distribution of the capital investment cost over the various years is illustrated. Expenditure in the first year is inevitably high, after which it decreases sharply, subsequently increases gradually and eventually falls again. It is evident that the investment cost has to be drawn solely out of capital during the first few years. After coal production commences, an operating profit from coal sales, which will cover part of the investment cost at an increasing rate, is realised. After some years this profit will be sufficient to cover the investment cost as well as the interest on capital.

The operating profit is therefore an essential factor, the influence of which will increase with increased capacity of production at the mine being developed. The larger the mine is planned, the greater is the proportion of the investment cost which can be covered by the operating profit. On a mine with a daily production of 4,000 tons, the operating profit will meet the interest on capital and may also cover up to 5 per cent. of the investment costs. The approximate corresponding percentages for mines of 4,000 and 6,000 tons per day are 10 per cent. and 20 per cent. to 30 per cent respectively. It is interesting to note that the sum drawn from capital on a mine having an output of 6,000 tons per day is not very much higher than for a 4,000 tons per day mine, since the development charges for mines of from 4,000 to 6,000 tons per day can be met largely from the operating profit.

(*f*) *Time required for mine development.* The development time for a new mine can be separated into three main periods. The first period will cover the time required from the commencement of operations until the start of coal production; the second period will include the development of the output up to the planned maximum; while in the third period, the remaining development, including auxiliary plant requirements and/or secondary shafts, is completed.

The length of the first period is dependent mainly on the depth of the shafts required, and includes surface preparation, which may take six months, and shaft sinking, which may require from two to four years. The construction of shaft insets and the first drivages in the rock will probably require from one to one and a half years. Thus the total time required, before regular coal production begins, may be as much as from three and a half to six years. For the second period of

development, approximately three years will be required for a 3,000 tons per day mine and five and six years for 4,000 and 6,000 tons per day collieries respectively. The total time required in the latter cases will therefore be from six and a half to nine years and from nine and a half to twelve years respectively. The time required to cover the third period will vary according to local conditions, especially with respect to auxiliary shafts. These figures are given as comparative time schedules, since the actual time required will vary according to the particular conditions, but the time schedule does indicate the period required to attain full production in each case.

(*g*) *The influence of area and output on the production cost per ton.* Production cost per ton of output includes many factors, or cost categories, of which wages and salaries, materials for support, machinery, supplies and other materials, general expenses and depreciation will be considered.

Wages costs depend upon the number of shifts required to be worked quite apart from rate of pay. The total number of shifts worked will normally increase with rising output, if a comparison is made between mines having outputs of 3,000, 4,000 and 6,000 tons per day. A proper basis for comparison, however, is to relate the total shifts and wages per ton, 100 tons or 1,000 tons of output.

The total shifts worked per ton of output will not vary greatly at the face or on development work, and only slightly on haulage operations. The slight increase in the number of shifts worked due to longer haulage distances can be compensated by the probable saving in shifts worked at the shaft bottom. The shifts worked per ton of output at the surface will show a considerable decrease when considering the higher output unit. On the whole, the mine with a daily production of 6,000 tons will show a slightly lower wages cost per ton than the 3,000 tons per day unit, and this difference will probably be 5 per cent. less. In the case of salaries cost, due to lower administration costs, the difference will be in favour of the larger mine to the extent of from 2 to 5 per cent.

Costs per ton for supports, machinery and other underground materials and supplies can be assumed to be the same.

The materials cost on the surface in the case of the larger mine will be approximately 25 per cent. less than at the smaller 3,000-ton unit.

When considering general expenses and depreciation, the 6,000-ton unit shows to advantage by as much as 10 per cent. on expenses and about 15 per cent. on depreciation costs.

The total production cost per ton of output within the areas discussed decreases with increased output. It can be stated, therefore, that a mine having a daily production of 6,000 tons will operate at a cost per ton of from 5 to 7 per cent. less than a mine planned to operate at a capacity of 3,000 tons per day, assuming the geological conditions are identical in each case.

(*h*) *Relationship between output and output per manshift.* Since the wages cost is a large proportion of the overall production cost per ton, the output per manshift attained affects the production cost to a great extent. It should be noted, however, that a large mine may be worked at an economic cost, even when the output per manshift is comparatively low, while a small mine has to achieve its economic operation within very narrow output limits. Fig. 64 compares the production costs and profit per ton related to varying values of output per manshift. Assuming the same working conditions and the same output per manshift, the mine producing 3,000 tons per day may work at a loss, while mines producing 4,000 and 6,000 tons per day are profitable undertakings. This is possibly due to the smaller influence of the cost per ton of interest on capital invested in the case of mines with high-output capacity.

(*i*) *Production cost per ton and rate of employment.* The previous deductions are all based on the ability to dispose of the output on a fair market, and that full employment exists at the mines in question. This is, however, not always the case. The coal-mining industry in every country has had to face periods in the past when coal sales dropped and the rate of employment and degree of mechanisation was decreased. Cost categories depending upon the rate of employment have a serious effect on the production cost—that is, depreciation and interest on capital investments. Since these charges remain the same whatever the level of production, this part of the cost per ton is inversely proportional to theoutput. Of the remaining cost categories, only a few decrease at the same rate as the output, while most decrease at a lower rate, thus effecting a rise in production cost per ton. The 6,000 tons per day mine will need to anticipate at least the same production costs per ton as the 4,000 tons per day mine, if production falls to that level—that is, if the rate of employment falls to 66 per

cent. If it drops to 50 per cent., the mine can expect to have at least production costs per ton equivalent to the 3,000-ton mine.

The degree to which a large or small mine, both operating under the same geological conditions and with similar technical equipment, is affected by a decreased rate of employment is mainly governed by the depreciation cost per ton. In the case of the larger mine the depreciation cost per ton is lower than in the case of the smaller

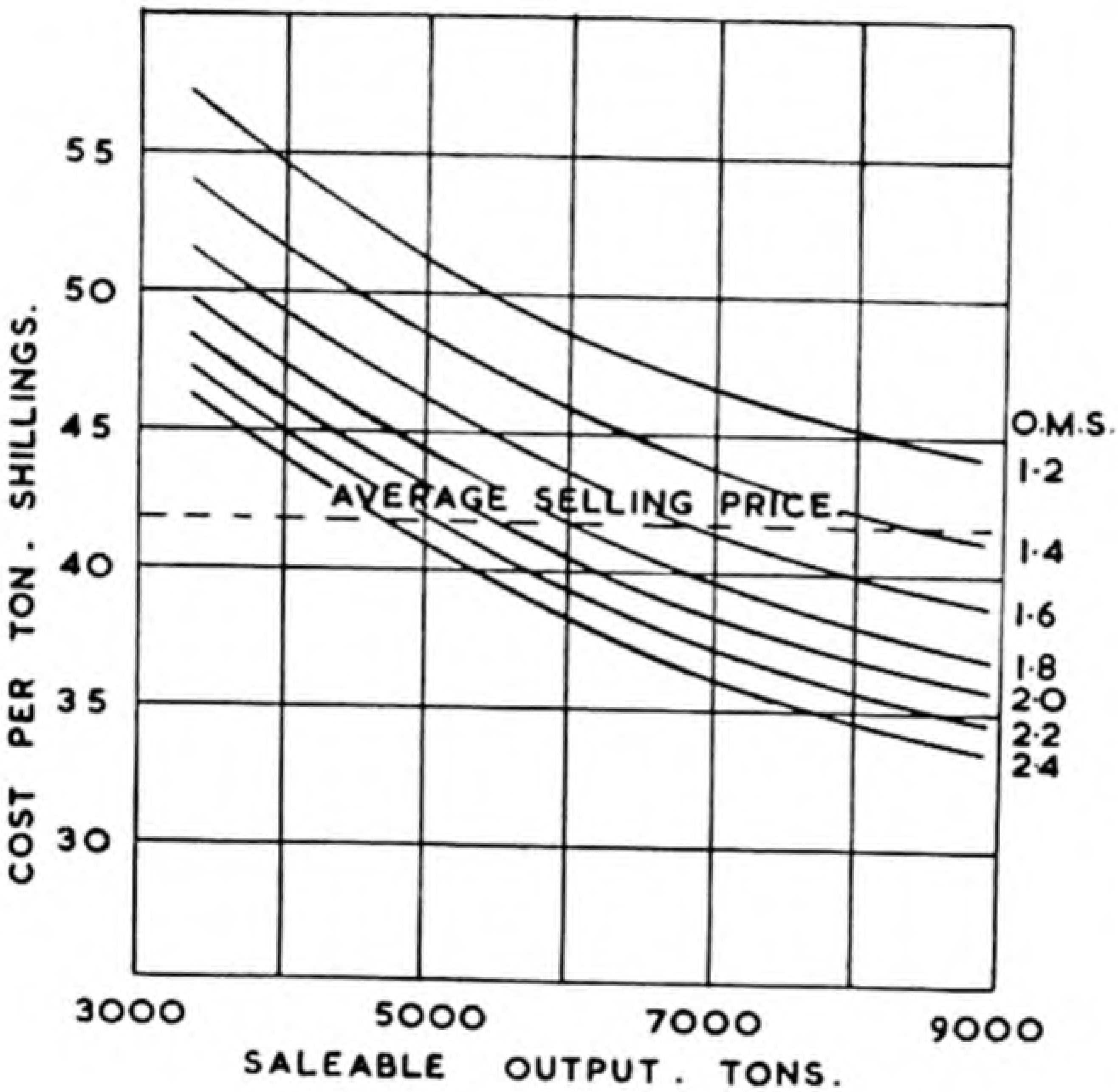


FIG. 64

mine, and the superiority of the larger mine is more evident if the invested capital has already been written off.

(*j*) *The optimum output of a mine.* A mine can be said to have attained its optimum output if production costs are at a 'minimum'. In the examples quoted, the optimum was assumed to be 3,000 tons per day for an area of 1,500 acres with coal reserves of 60,000 tons per acre, or 6,000 tons per day for an area of 3,000 acres. It should be noted, however, that this optimum, under different geological and technical conditions, may be much less or even greater, say 2,000 or 4,000 tons for an area of 1,500 acres. It should also be remembered that the increase of output per shaft, by extending the area from 2,000 to 3,000 acres, has its limitations. It would be impracticable to in-

crease the output per day to 12,000 tons by extending the area to, say, 3,000 acres. Technical difficulties would increase and eventually become prohibitive, if only by exceeding the winding capacity of the shafts. Since production costs are also affected by such technical difficulties, the term 'minimum' cost in the original definition presumes that the production cost is higher if the optimum output for the minimum cost is exceeded or not reached. The curve rises sharply at first, followed by a gradual fall to a minimum and subsequently followed by a rise. The various factors affecting the rate of change in the level of the production cost can be analysed in detail, but only a few of the main factors need be considered at this juncture. The development costs for the main shafts, staple shafts and main roads are practically independent of the output level under the same conditions. The same applies to the installation of ventilation and drainage equipment, part of the officials' salaries and part of the general expenses. Thus, these costs are dispersed over a larger tonnage and, with increasing output, the cost per ton will fall.

The flattening of the graph, and the subsequent rise indicating a cost increase, can be attributed to various factors. Two factors which can be discussed are main road maintenance and ventilation. An increase in output will demand an increase in the quantity of air to be circulated and therefore the water-gauge required. Due to the greater pressure, it can be anticipated that the normal leakage will be increased and a greater power cost ensue. There is, however, a restriction on the quantity which can be passed in any airway. The greater number of working faces required, the greater is the amount of roadway maintenance and volume of air necessary for ventilation. The increased length of roadway not only increases maintenance cost per ton, but decreases the flexibility of extraction operations, and makes the preparation of stand-by faces more difficult.

The optimum output has to be carefully worked for every mine area. An area containing large coal reserves per acre will have a higher optimum output than the same area with lower reserves. The occurence of irregular geological features and the presence of thin seams will reduce the optimum. Where mines have a high methane emission per ton, the optimum output would be reduced from what would have been anticipated under otherwise equal conditions, and the incidence of heavy strata pressure will have a similar effect.

(*k*) *Output capacity of a mine.* The output capacity of a mine may

be defined as the highest possible output. This output not only depends upon the natural conditions, as assumed in the previous deductions, but also on the efficiency of technical equipment. The efficiency of the gate road haulage, staple-shaft winding, main road haulage and main-shaft winding are all contributory factors. The efficiency of the technical organisation and equipment of the mine should correspond with the output capacity, if natural conditions are assumed. If the technical equipment has been installed under capacity, the available output capacity of the mine is not fully utilised, the equipment is overloaded and the cost per ton is increased. If, on the other hand, over-estimation has taken place in the purchase and installation of equipment, unnecessary capital expenditure has been made and the depreciation cost per ton will increase unnecessarily.

It is also essential that the efficiency of the various units of the equipment and machinery installed should be well balanced. The efficiency of face conveying should compare favourably with that of gate road haulage which, in turn, should be equally capable of satisfying staple-shaft capacity. The same argument applies to other stages of haulage. While over-powering should be avoided, sufficient reserve should be allowed in the equipment to meet temporary breakdowns and peak loads.

Section 10. *Development and Maintenance Costs in the Horizon Mining System*

(*a*) *Comparison between fully developed mines and those in course of development.* As has been previously explained, all main haulage roads and staple shafts are usually in stone, if the horizon mining system is applied, while gate roads are driven in coal.

The number and length of main roads to be driven in a new mine are, as a rule, twice as great as for an older mine which has already been developed. In a new mine two levels have to be driven, viz. the ventilation level and the first haulage level. In the initial stages, this development is advanced far enough to enable the required number of working points to be opened out. While coal production is maintained between the two levels, development work is continued and eventually completed. A stage is reached when the coal above the first haulage level is exhausted, and this level becomes the ventilation level for the next haulage level. From this time, only the new haulage level will have to be developed and the number of cross-cuts and lateral

roads will be reduced by about half. The vertical development—that is, the number of staple shafts—will be the same, unless the geological conditions are such that a number of shafts are required of different depths.

The number and length of gate roads driven in coal will not change, whether coal is mined between the first and second level, or between two deeper levels. There will be a difference if the gate roads on a haulage level have to be maintained for use as ventilation roads for the next haulage level, but, as a rule, this very seldom arises.

(*b*) *Driving costs of the roadway grid in stone*. The total cost for the driving of the main road grid depends upon the two main factors: total length of main roads and driving cost per unit length.

The total length of the main road grid in a concession of a given extent is affected by the number of cross-cuts required—that is, their distance apart—and by the number of lateral roads. One lateral may not be sufficient if the concession is wide across the strike of the measures.

In the example shown in Fig. 62, three cross-cuts and one lateral road are assumed for every level. The length of the mining area, or concession, is 4,320 yards and the width 2,185 yards. The length of each district cross-cut is therefore also 2,185 yards. As the distance between cross-cuts is approximately 1,440 yards, the length of the lateral road will be 2,880 yards. The main road grid is therefore $3 \times 2{,}185 + 2{,}880 = 9{,}435$ yards on one level, or 18,870 yards on two levels, excluding the shaft bottom.

Staple shafts are also part of the main road grid. Since the mining area is to be split up into ten working panels, each approximately 200 yards wide, a maximum of eleven staple shafts will be required for each district cross-cut, i.e. a total of thirty-three staple shafts. With a distance between horizons of 200 yards, the total length of staple shafts will be 6,600 yards, if they extend from level to level, or 5,500 yards if every second staple shaft extends only to the top seam. Additional items of development work are the pit bottom, workshops, store, pump-rooms, water standage and the sinking of at least two shafts each 250 yards deep. In order to assess the total cost, the drivage costs per yard for roads, staple shafts, etc., have to be ascertained, bearing in mind the probability of different sectional areas for main roads and cross-cuts as well as main haulage and ventilation levels, which will depend upon their position and purpose.

		per yd.
250 yards pit bottom	. . . .	at £150=£ 38,000
2,880 " lateral roads	. . . .	at £ 60=£173,000
6,555 " cross-cuts	. . . .	at £ 50=£328,000
5,500 " staple shafts	. . . .	at £ 60=£330,000
By passes and staple shaft bottoms	. . . .	=£ 25,000
Pump rooms, water standage and other rooms	.	=£ 18,000
500 yards main shaft	. . . .	at £350=£166,000

As the coal reserves between two levels are 19·2 million tons, the driving cost of the main road grid per ton is about 13·5*d.* (including the main shaft).

Cost of the ventilation level is not considered, as this has already been included in the initial development cost and need not be introduced again at this stage. There would be 2,880 yards of lateral road and 6,555 yards of cross-cuts, the drivage cost being less by, say, 20 per cent. than on the main haulage level, due to the smaller cross-section. The drivage cost of the ventilation level will be £430,000, or 5·4*d.* per ton, which is a comparatively low figure. In a mine area where geological conditions are less regular, a greater number of cross-cuts and staple shafts may increase the cost by 50 per cent. or more. The same argument applies to the case where the reserves between two levels are smaller. If the coal reserves were only about 10 million tons instead of 19·2 million tons, the effect would be to double the cost per ton of the main road grid of the haulage horizon to 2*s.* 2*d.* Conversely, a higher reserve figure would reduce the cost per ton.

There is obviously a maximum drivage cost per ton beyond which the economic working of the mine is seriously affected. This upper limit cannot generally be fixed, as it depends upon the other relative costs and the profits in each individual case. If the conventional dip-mining system can be used, the saving effected by the adoption of the horizon system should be carefully considered, taking into account a similar cost computation for the dip-mining system in each case.

In order to illustrate the scale of stone drivage costs, it may be noted that these costs vary greatly in flat formations in the Ruhr mines, i.e. between 4 and 12 per cent. of the production cost per ton. This would mean 3*s.* to 8*s.* per ton, with production costs at £3 10*s.* In the Aachen hard-coal district development costs are

9–12 per cent. of the production cost, and in the Belgian Campine 6–8 per cent.

According to these costs, in a Ruhr mine, working in flat formations, from 2 to 5 feet of main roads have to be driven in rock, and from 6 inches to 2 feet of staple shafts have to be sunk per 1,000 tons per annum. Thus, with an output of 1 million tons per annum, 2,000 feet or 5,000 feet (700–1,700 yards) of main roads are driven, while 200–700 yards of staple shafts are sunk per annum.

In mines working steep formations, the total length of roads per 1,000 tons per annum is usually slightly greater than in those working flat seams, since longer cross-cuts have to be driven on one or several intermediate levels. Sub-level cross-cuts from staple shafts are also found in flat formations, in particular in the vicinity of faults, but they are usually short in length. The distance between district cross-cuts is often slightly less in steep formations. Since belt conveyors cannot be used for gate conveying in steep seams, winches and small locomotives have been used, but their capacity decreases with increasing length of the gate roads.

(*c*) *Driving costs of roads in coal* (*gate roads*). The total cost of drivage will again depend upon the number and length of gate roads required, as well as on the cost per yard.

The number of gate roads is closely connected with the length of faces developed. Where short faces are won out, a considerably greater number of gate roads is required than in a mine of equivalent output where long faces are developed. In a seam in which ten faces, each 120 yards, can be developed (measured on the dip), the same total length of working face can be obtained with five faces, each 240 yards long.

In the first case eleven gate roads are required and in the second case only six gate roads are necessary, assuming that the coal-conveying road of the upper face will be used again as the ventilation road for the next lower face. This latter method of development is often impossible, so that a new road has to be driven. Since this applies to both cases, the relative difference still remains.

The total length of the gate roads and the length per 1,000 tons per annum may be calculated for the example shown in Fig. 62. Assuming a daily output of 4,000 tons, at least six double-unit faces are required, advancing five feet per day. Three gate roads are required for each double-unit face, and have to be advanced 5×250 feet, or

about 415 yards per annum (assuming 250 working days). Every double-unit face, therefore, requires about 3×415, or approximately 1,250 yards of gate roads per year. For six double-unit faces the total length of gate roads to be driven is 7,500 yards. This would give from 7 to 8 yards per 1,000 tons of output per annum, which is a fair figure for mines in flat formations. The average in the Ruhr district is from 5 to 8 yards. As the life of the level considered is nineteen years, the calculated total length of gate roads to be driven in that time would be 142,500 yards. This length would actually be smaller, however, since some roads would be used jointly by two adjoining double-unit faces.

The cost for gate roads in the example quoted is $7{,}500 \times$ £13 or £97,500, if the total driving cost per yard is assumed as £13, which is a cost per ton of 2*s*. In steep formations the total length of gate roads to be driven is usually greater than in flat formations. In steep seams the length of the faces is reduced so that more gate roads have to be driven. The length per 1,000 tons per annum will be from 8 to 15 yards or more.

(*d*) *Maintenance costs of stone roads.* The maintenance cost for stone roads and staple shafts depends upon their total length, and this cost usually is expressed as the average cost per 100 yards of road or shaft. The total length of stone roads to be maintained is at its minimum when the full production on a new level begins, as only part of that level has then been driven. The total length is a maximum when production from the level is just coming to an end, as almost all roads have to be maintained and development of the next haulage level is also in operation. In the example, Figs 65A, B, C, the lateral road is 2,640 yards long, and each cross-cut is at least 350 yards, so that about 3,700 yards have to be ready for the beginning of full production. In addition, there are the stone roads of the ventilation level, about 9,400 yards, so that there are about 13,000 yards of stone roads to be maintained.

At the end of a haulage level this length is increased by the side extensions of the cross-cuts, or another 5,900 yards, making the total 18,900 yards. In addition to this drivage, development on a new haulage level has commenced, so that a maximum length of about 20,000 yards has to be maintained, if only for a short time. As the total length, however, is only 13,000 yards at the beginning, an average length of 16,500 yards, or 16·5 yards per 1,000 tons per annum, can

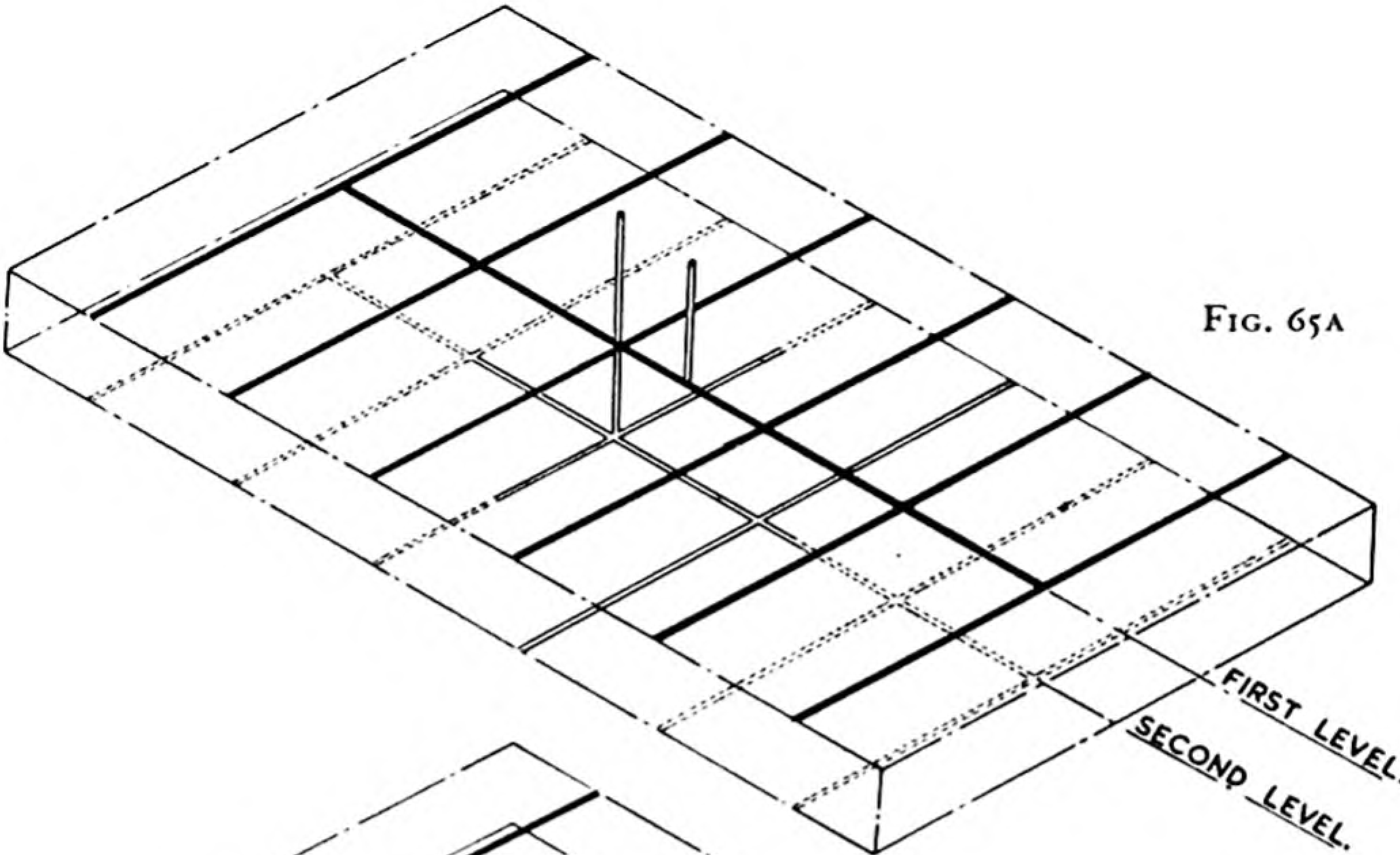


Fig. 65A

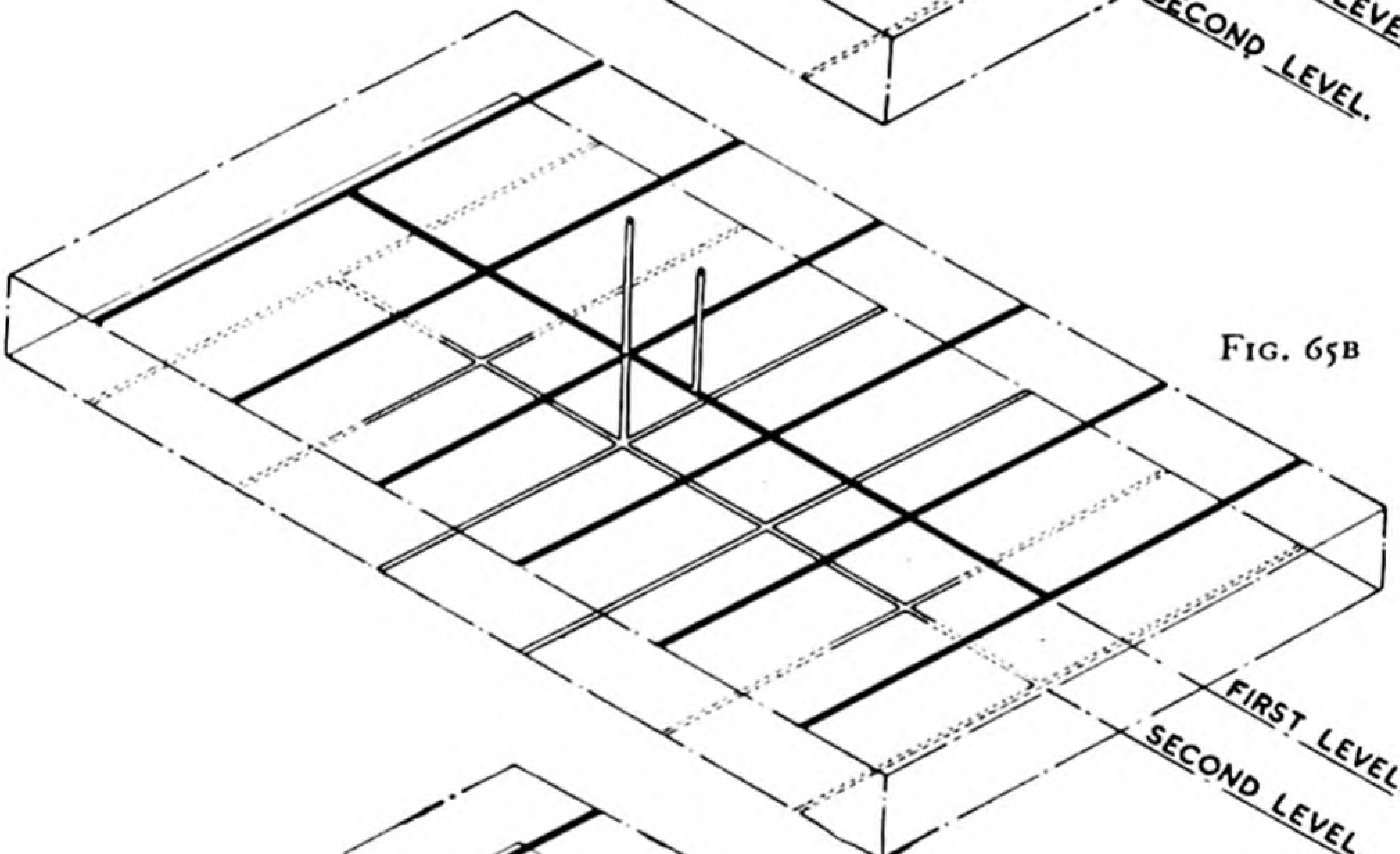


Fig. 65B

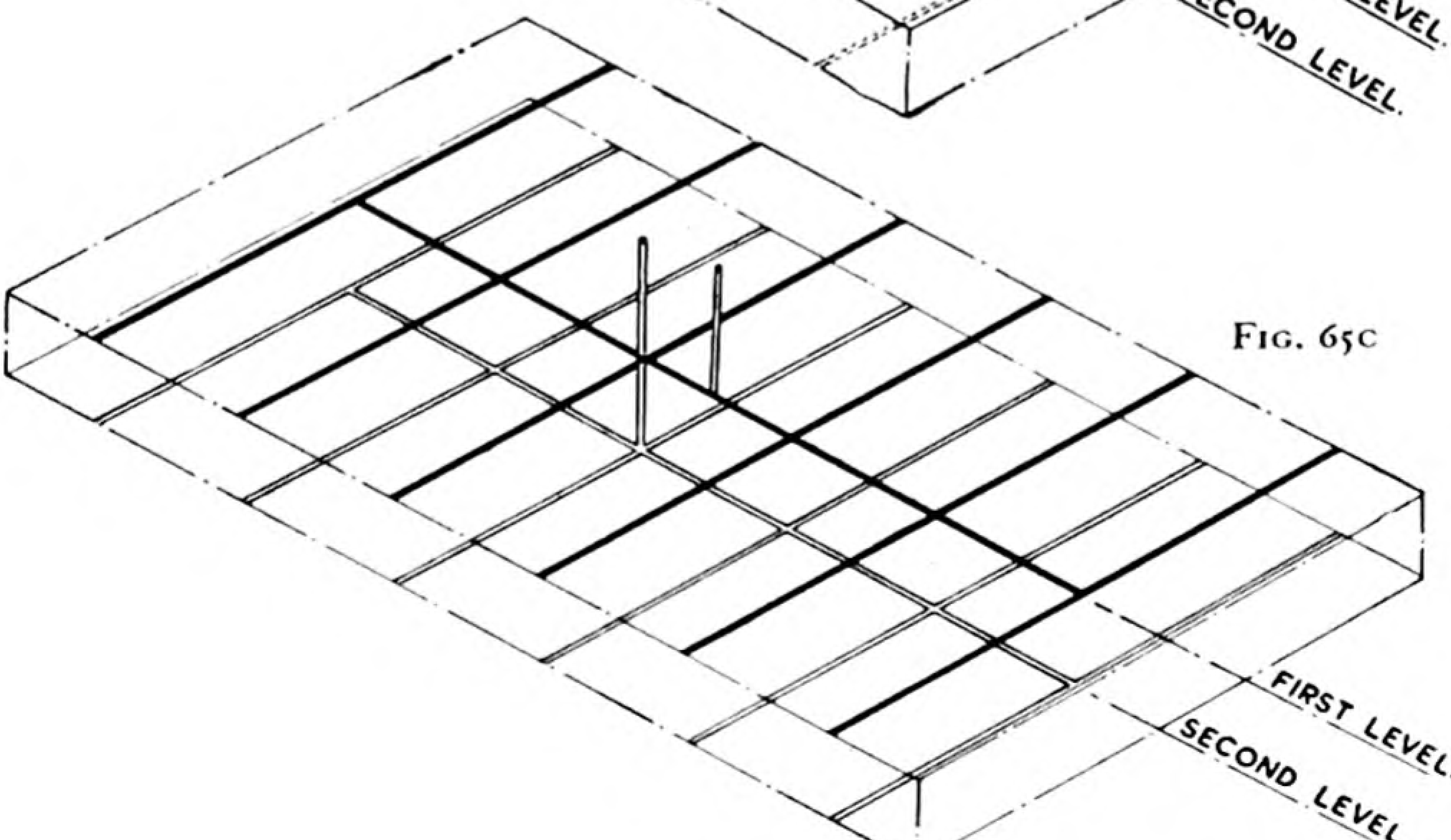


Fig. 65C

be assumed. In the Ruhr mines working flat formations this value is between 10 and 20 yards. In mines in steep measures the grid of roads is more extensive, due to the necessity for long intermediate levels; average figures under these conditions in the Ruhr are from 20 to 40 yards and more per 1,000 tons per annum.

Maintenance costs of stone roads depend essentially upon the type of rock in which they are driven. Roads driven in shale very often cost twice as much as roads in the stronger sandstone, while roads in sandy shales are intermediate between these values. The most suitable form of support is also very important, it being considered more economical to insert a stronger and more expensive form of support in the first instance than a less expensive type which requires frequent maintenance. The disposition of the various faces, the type of packing employed and the distance of the seams from the stone roads are all factors which have an influence on the maintenance cost.

Figures quoted for the Ruhr for roadway maintenance are 3 to 7 shifts per 100 tons, or 7 to 17 shifts per day per 1,000 yards of roadway. With an average figure of 5 shifts per 100 tons and wages at 30*s.* per day, the wages cost would be £7 10*s.* per 100 tons. Taking materials cost at 50 per cent. wages cost, the total cost would be about £11 per 100 tons or 2*s.* 2*d.* per ton.

(*e*) *Maintenance costs of staple shafts.* The number and depths of the staple shafts required again influence the maintenance costs, together with the type of rock, form of support and to what extent they are affected by the extraction of the adjacent coal.

In the example in Fig. 62, nine staple shafts are in operation when full production begins. The total number of shafts to be sunk is thirty-three, but they are not, however, all working at the same time. About eighteen shafts can be assumed to be in use simultaneously, but thirty-three shafts have to be maintained. The total depth of 3,000 yards corresponds to a length of 3 yards per 1,000 tons per annum. In the Ruhr, in mines working flat seams, 1 to 3·5 yards per 1,000 tons per annum are maintained, or 1,000 to 3,500 yards per million tons of annual output. The figure of 3 yards in the example quoted is comparatively high, due to the great distance of 200 yards between two levels.

For the maintenance of staple shafts, 1 to 2 shifts per 100 tons are required, or 40 to 80 shifts at a mine having a daily output of 4,000 tons. For every 100 yards of staple shafts, 2 to 5 shifts per day are

necessary. As 3,000 yards have to be maintained in the example given, 75 shifts per day or almost 2 shifts per 100 tons are required. With day wages at 30*s*., the wages cost is 60*s*. per 100 tons. Taking the materials cost at 33 per cent. of the wages cost, the total cost is 80*s*. per 100 tons, or 10*d*. per ton. This cost may be reduced with favourable conditions, or exceed 1*s*. per ton where there are adverse conditions.

Section 11. *Deviations from the Normal Layout*

(*a*) *General introduction.* In previous sections the horizon mining system has been discussed in its normal form of development, this being the general development in most coal-mining districts on the Continent. There are, however, deviations from the normal layout, some of which become systems intermediate between horizon mining and dip mining. The variation in the natural working conditions makes it difficult to apply a similar system of working in every case. The normal system will be modified to suit local conditions in order to give the best method possible. One such deviation is the so-called 'under-level' extraction, which has already been referred to in Chapter 1, Part II, Section 5 (*b*). Another possible deviation can be applied where the roads in one of the upper, worked-out seams, are used for ventilation roads. A third deviation from the normal method is represented by a combination between horizon and dip mining.

(*b*) *Under-level extraction.* In the normal horizon mining system, extraction is carried out above the haulage level only, the coal being lowered to that level and hauled to the shaft. With under-level extraction, the seams are mined below the haulage horizon and the coal raised to the haulage level.

This system is used in some cases when it is imperative to adopt the method because the normal development would be too expensive, due to the great distance to the isolated seams from a haulage level at a lower horizon. In the case illustrated in Fig. 66A, parts of the seam *A* could be developed from a long cross-cut and a deep staple shaft, from the lower haulage level, but it is easier and more economical to sink a short staple shaft from the ventilation level near the boundary. The small syncline *B* would require an unnecessarily deep shaft from the haulage level, while a short staple shaft from the upper level is sufficient. It is better, however, to connect the seam at *C* with

a staple shaft, which is necessary in any case for the extraction of the lower seams.

If a mine has been developed to its lowest horizon, there may still be reserves below that level which should be worked, but the quantity remaining may be insufficient to justify the development of a new haulage level at a lower horizon. In this case, under-level extraction is preferable, even to the extent shown in Fig. 66A. Similarly, if the development of the next lower level has been retarded due to lack of time or unforeseen difficulties in production, under-level extraction is one way to overcome this problem. It should be remembered, however, that the greater the extent of the development of under-level extraction, the higher will be the cost of production, and delay in main development should be speeded up as soon as possible.

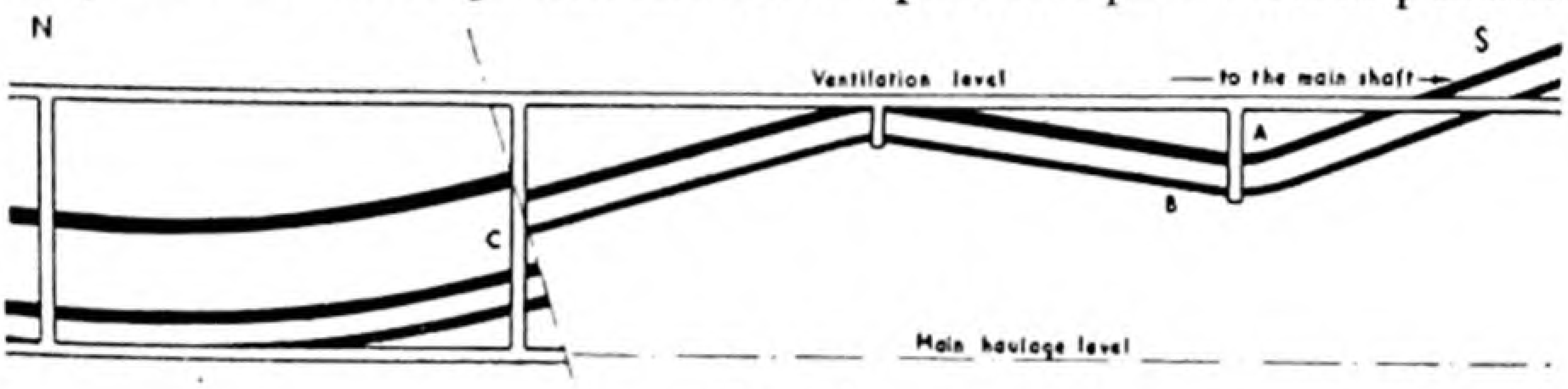


Fig. 66A

In all cases where under-level extraction is carried out, ventilation difficulties will have to be overcome, since the ventilation is descensional and the intake air from the haulage level has to circulate either through a staple shaft or a dip heading to the working faces. The return air should be passed to the return air level by the shortest possible route. Short-circuiting of the air must be avoided, but as a rule the negative pressure in the return horizon will be sufficient to prevent a complete short-circuit.

In addition to ventilation, the drainage has to be carefully considered. In under-level extraction, the working area is the deepest section of the mine and special drainage arrangements must be provided if the strata is water-bearing.

The greatest disadvantage of this system of extraction is the damage which is likely to result to the main haulage level from the mining operations underneath. The main roads will subside, since the lateral pressure produced by the under-level extraction affects the surrounding strata, resulting in expensive maintenance work to both roadway supports and haulage track. If the lower seams are gassy, it is probable that gas will be directed into the main haulage roadway

through crevices and fissures in the strata, causing a high gas concentration and creating a general ventilation problem. Because of these disadvantages, under-level extraction is employed only where conditions make it imperative. Where this system is adopted, the haulage level will deal with coal worked above and below the level, as shown in Fig. 66B. This example illustrates that the second level

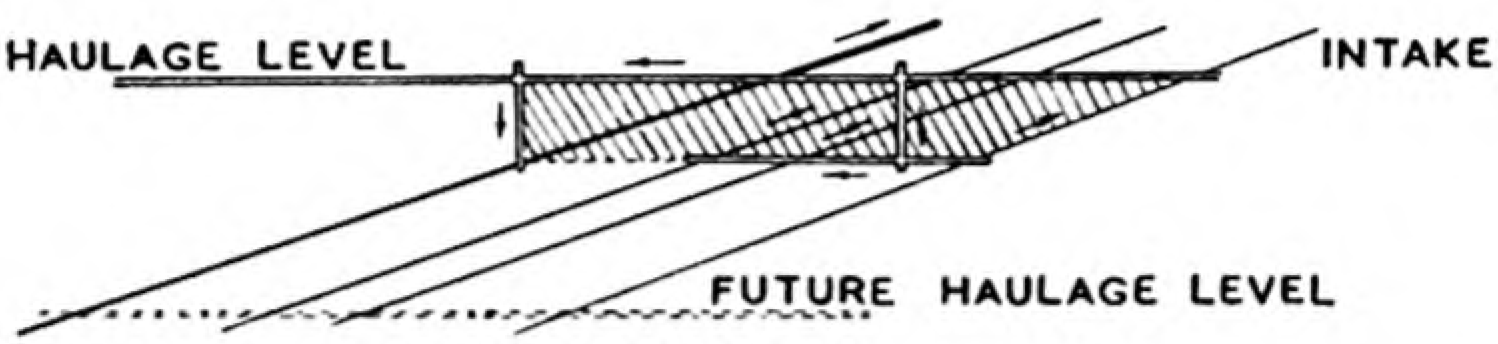


FIG. 66B

acts as a haulage level for the coal reserves above and also for the coal mined in the upper part of the third level, this coal being worked by under-level extraction from the second level. By adopting this method, the life of the second level is increased, and more working points are made available, with a consequent increase in productivity, The more extensive the under-level extraction has been from the upper level, the greater can be the horizon interval to the next haulage level. If the downcast shaft has been sunk down to the next level, normal ventilation can be applied and intake air can be directed from the lower level to the under-level extraction area. In this scheme of development, shown in Fig. 67, ventilation and drainage problems

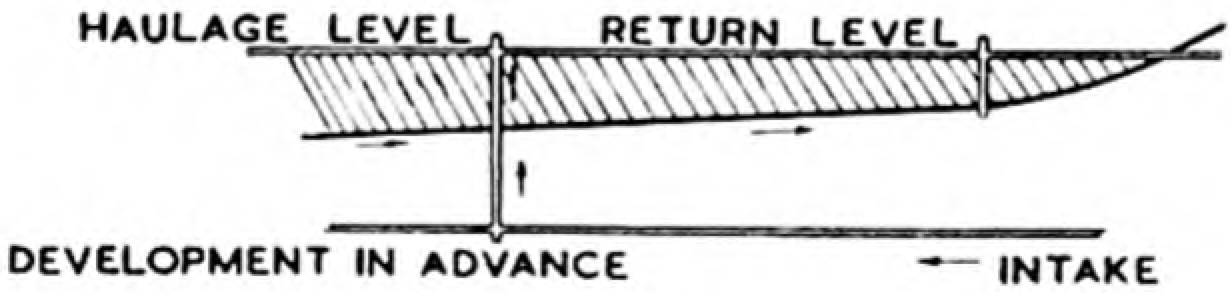


FIG. 67

would not necessarily arise, but damage to the haulage level may still take place. Quite apart from these considerations, systematic under-level extraction requires the drivage of a greater number of roads in stone for the numerous intermediate levels, and its adoption should be limited to exceptional cases.

(*c*) *Roads in an upper worked-out seam acting as a return air horizon.* Where the conventional mining system of driving roadways in the seam has been in use and the development and extraction of lower

seams is to be carried out by horizon mining principles, the roads in the immediate upper seam may be used as a return air horizon. The main haulage horizon will be the only level required to be driven in the strata. Some development work in driving goaf drifts will be necessary in the return air horizon, in order to connect the top of the staple shafts to the nearest roadway at this level. In general, the old roadways existing in the return air horizon will not correspond exactly in position to the cross-cuts and laterals of the main haulage level. Several cases in which this method of development has been adopted in British collieries are described and illustrated in Appendix 1.

(*d*) *Combined horizon and dip mining system*. Certain geological and mining conditions permit the use of a combination of the normal dip system of working with horizon mining development. The combination takes advantage of the level locomotive haulage roadway and of belt conveying in the dip-coal headings direct from the winning faces to the main haulage roadway. Two cases which have been introduced at British collieries are illustrated and described in Appendix 1. The system reduces the initial development required by the normal horizon mining system by cutting out numerous staple shafts. In many cases advantage can be taken of existing roadways in upper seams to act as a return airway level. The first case illustrated in Appendix 1 utilises dip-coal headings, driven from an upper horizon cross-cut, to develop diverging longwall advancing faces, while in projected developments at colliery 'G', the coal headings will be driven to the rise to form diverging single-unit faces. The locomotive roads will be driven in the seam, following the level course of the seams in the coal basin forming the area to be developed.

Section 12. The Advantages and Disadvantages of the Horizon Mining System

The main advantage gained in adopting a horizon system of mining and development is in the greater flexibility available in the choice of a main-road haulage system. Since the main roads are level, locomotive haulage can be considered. Continental experience has shown that, under their conditions where horizon mining is practised, locomotive haulage is the most economical and effective method of coal haulage underground. This method is even more

favourable where stowage material, supports and other supplies have to be brought to the face. When considering rope or locomotive haulage, the manshifts required per 100 tons of coal conveyed may be compared. For rope haulage in the Ruhr the average is 10 manshifts per 100 tons and for locomotive haulage 2 manshifts per 100 tons. Where man-riding is necessary it is easily accomplished with locomotive haulage without additional equipment: with rope haulage, special man-riding equipment should be used and it may be necessary to install an additional haulage engine for this purpose in another road. It is generally agreed that under similar conditions locomotive haulage is more efficient than rope haulage, although there are many cases of high-capacity rope haulage systems in operation. The flexibility of the locomotive haulage system, however, makes it far superior for the transportation of large quantities, say 4,000 tons or more, to a single point during two shifts and allows quick and punctual return of empty mine cars to the loading-points inbye. Any alteration in the capacity of the main haulage can be dealt with easily in a locomotive haulage system by reducing or increasing the locomotives or number of mine cars in use.

With the horizon mining system, ventilation can be more efficient, as the main intake and return airways are completely separated and, since it is not necessary to provide air-crossings at frequent intervals, leakage is minimised. These beneficial results make it possible to ventilate coal faces farther from the shaft more effectively than would be possible under the same total quantity conditions in the conventional system. It should also be noted that with the horizon mining system the intake air is cooler on reaching the face, since the intake airway is in rock which has a cooling effect varying with the type of strata through which the drift is driven. Coal sides act in a like manner but to a very much less degree, as the thermal conductivity of coal is higher than that of rock, while pack walls have a still higher thermal conductivity. This advantage is greater, the greater the depth of the mine and the higher the working temperature. A further advantage is that the intake air is kept almost free from methane, since the roadway is in stone.

With the conventional system where the roads are in coal, the intake air absorbs a certain proportion of methane before the working face is reached.

The horizon mining system allows the possibility of working

several seams simultaneously. Thus, a mine developed on this system can balance its output according to the quality of coal required for processing or other market needs. With the conventional system of mining, two or more seams may be worked at the same time, but in this case it is necessary to introduce phasing of the extraction, which is not as simple or as economic as in the horizon mining system. The main disadvantage of the horizon mining system is the greater initial cost brought about by the driving of expensive drifts in rock and the longer time required for initial development before coal-winning is commenced. In the conventional system, coal working may be commenced as soon as the shaft reaches the seam, while in horizon mining, development in rock of thousands of yards of stone drift and the sinking of several staple shafts is required.

Horizon mining also requires a certain minimum reserve of coal available, varying with conditions but sufficient to amortise the cost of stone drifting and cover interest charges. This minimum reserve varies according to the nett proceeds, capital expenditure required and the geological conditions, and may be estimated at approximately 10 million tons per level.

CHAPTER 2
STRATA CONTROL AND SUBSIDENCE

PART I

THE GENERAL PRINCIPLES OF SUBSIDENCE

Section 1. *Strata Movements above Working Areas*

THE state of equilibrium of the strata is immediately upset as soon as any excavation is made, this change being due to the inability of the strata immediately above the excavation to maintain the weight of the superincumbent beds. These zones, therefore, have to sustain an additional load, apart from the weight of the strata they previously carried before the excavation was made.

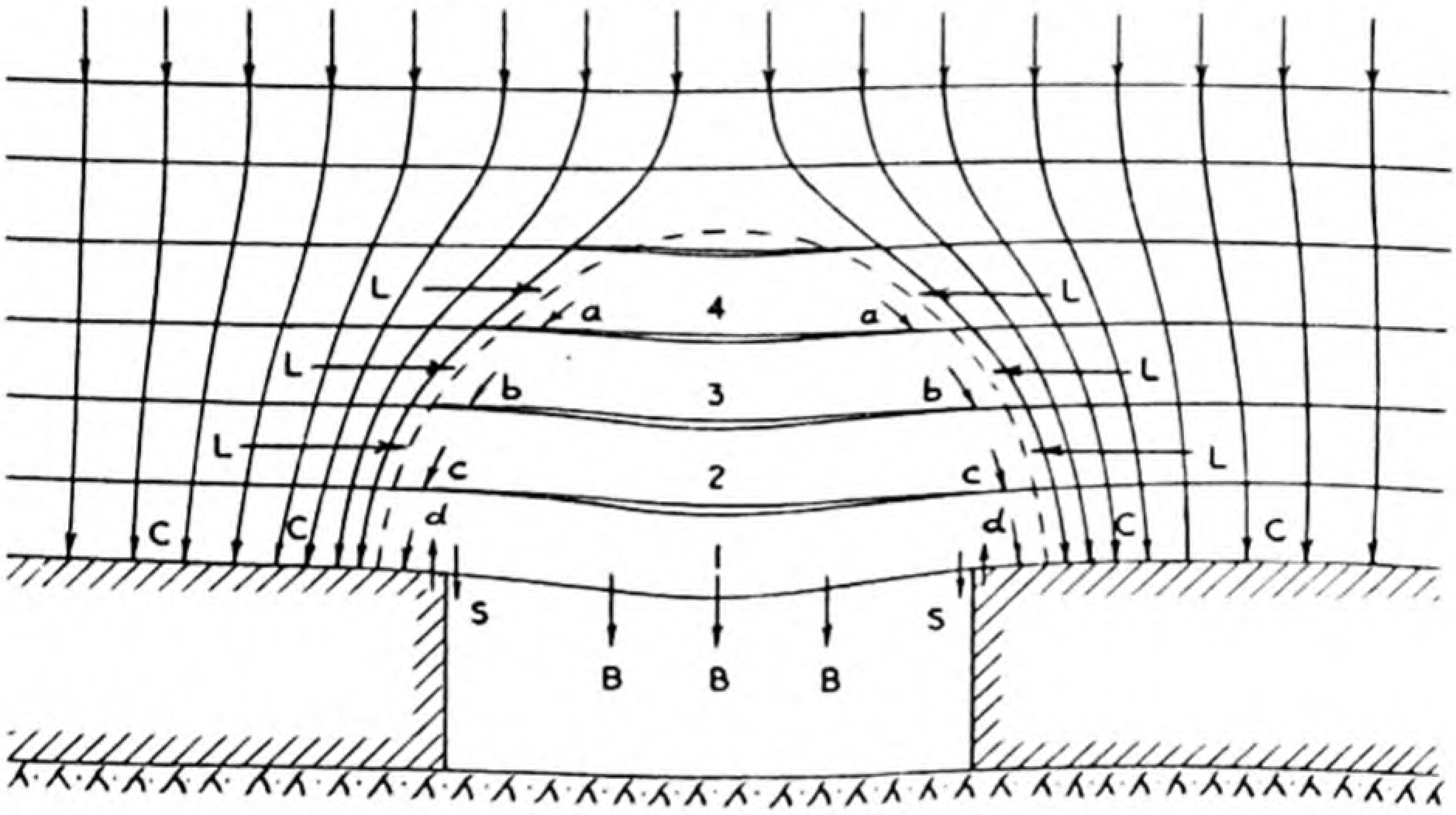
The downward bending of the beds, *vide* Fig. 68, temporarily frees them of the weight of the beds above, and the weight of the strata above the roadway is transferred to the solid coal at the sides of the excavation.

The redistribution of the forces previously acting is continued, other forces being induced until a state of equilibrium is restored. The influence of these forces on the roof strata is indicated in Fig. 68, in which the action of shearing, compressive and bending forces is illustrated. The transference of the load, or weight of the strata, to the border zones, or 'abutments', results in the formation of a 'pressure arch', within which there is a zone of strata relieved of stress. These zones of stress concentration, or abutment zones, may be beneficial in assisting coal-getting at the face, or create adverse conditions in gateways, cross-cuts, etc., in which the effect of strata movement and the presence of abutment loading is to cause disruption.

The effect of the lateral compressive forces, the intensity of which will increase with depth, acting along the roof strata is to attempt to regain the natural balance by causing the strata to expand into the excavation from over the solid. If the excavation is small, or if the roof strata are very strong, the cavity may remain open for a considerable time. This condition does not arise, however, as the worked-out areas in coal seams are normally very extensive, and

the overlying strata are not strong enough to remain unaffected by the extraction. The immediate roof bends and bed separation and subsidence of the strata take place, finally breaking, and the effect is carried to the upper strata beds. The cumulative effect of this subsidence finally reaches the surface.

In such a case the influence of the stowage or packing method employed during the extraction is to limit the extent and nature of



B BENDING FORCES L LATERAL COMPRESSIVE FORCES
S SHEARING FORCES C VERTICAL COMPRESSIVE FORCES

FIG. 68

the resulting subsidence but not to eliminate it altogether. The influence of the redistributed forces acting on all sides of the excavation is governed by the depth and the strength of the coal, and roof and floor strata. The rate of subsidence of the immediate roof beds will also depend upon the nature of the overlying strata; clayey shales subsiding faster than sandy shales, while the latter, as a rule, subside faster than compact sandstones, which may bridge the excavation, acting as beams and remaining in this condition for some time before finally breaking. Subsequently, the upper overlying strata will subside, carrying the effect to the surface. The interaction between two different types of strata, such as sandstone and shale, may delay the subsidence, due to the lower rate of subsidence of the higher sandstone beds, but will not stop the subsidence.

Flat cavities within the strata will be compressed by further subsidence of the overlying strata, and the support offered by the fallen and broken beds may tend to maintain the upper beds, which act as beams. The probability of such a stage being reached, in which the subsidence does not reach the surface but ceases at an intermediate bed, is very remote, being dependent upon the area of the excavation, which is again related to the depth. Such an area would be extremely small, and unusual, if the coal is systematically extracted. Thus, subsidence usually extends to the surface.

It is now well known that the subsidence does not extend vertically above the excavation to the surface, but affects a zone greater than the area of excavation, such that the zone extends outwards and upwards from the worked area at an angle of less than 90 degrees from the horizontal. At the surface, a subsidence trough is formed which extends beyond the worked area, flattens out towards the sides and finally levels off to the existing surface level. The conception upon which practice in the Ruhr is based can be described as follows: Assuming a rectangular worked-out area, the strata affected by subsidence take the form of an inverted obtuse pyramid, *vide* Fig. 69. The angle between the sides of the pyramid and the horizontal is defined as the 'angle of draw' (in British practice, draw is denoted as the angle to the vertical), and denotes the lateral limit beyond which the working has no measurable effect. The 'angle of fracture' is the angle at which the strata break, or fracture, especially if sandstone is predominant or the seam is thick. It is as a rule steeper than the angle of draw. Thus, the angle of fracture may be defined as the limit of maximum tension at the surface and along the plane of fracture and does not, necessarily, imply actual fracturing of the strata. If shales are predominant in the strata, the beds will bend and conform to the subsidence trough. The more plastic and soft the strata are, as with a quicksand overburden, the flatter will be the angle of draw, the larger will be the difference between this angle and the angle of fracture, and the greater will be the extent of the affected area on the surface.

Within the central pyramid affected by the subsidence the disturbed strata will move downwards under their own weight or gravitational load, tending to fill the excavation. The strata will also move laterally into the affected area. These lateral movements are a necessary condition for any subsidence outside the theoretical prism com-

prising the zone immediately above the excavation, since, without such movement, the strata would not flow from the boundaries towards the interior of the affected zone and move downwards to fill and consolidate the excavated area.

These vertical and lateral movements on the edge of the affected area set up tensile and compressive forces, as has already been described with reference to Fig. 69. Tension effects will occur at the boundaries of the affected pyramid, while compression effects, due to the increase in pressure, exist in the interior. In addition to these tensile and compressive forces which act laterally through the disturbed zone, there are vertical tensile and compressive forces present which

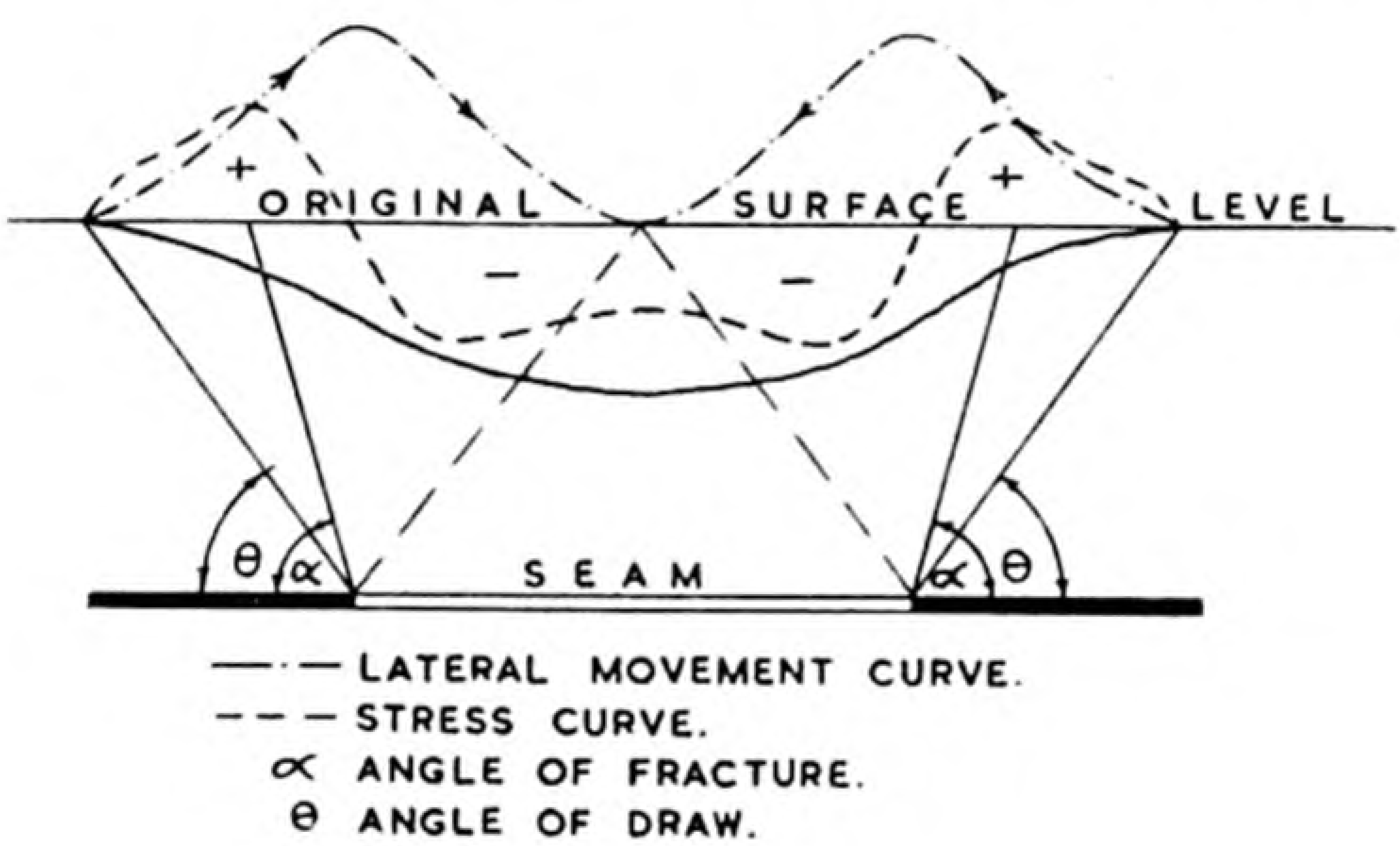


FIG. 69

have an additional effect on the support of underground roadways and on main shafts and staple shafts. Surface lateral movement, however, results only from the presence of horizontal tensile and compressive forces.

Section 2. The Angle of Draw and Angle of Fracture

These angles are not at all constant, being dependent upon the character and dip of the strata.

The angles are steep as a rule in strong strata, and are very flat in running sands which is a common feature in Western Germany, Holland and Poland. There have been exceptions to this rule, and steep angles have been recorded in loose overlying beds. The variation of the angles with the dip is important, the angles to the dip and to the rise side of the extracted area, where the strata are inclined, being

quite different. The angles at the sides of the extraction area are also influenced by the dip.

The following values for the angle of fracture have been accepted after long observations in the Ruhr:

In carboniferous strata,

(*a*) To the dip side,

Dip	*Angle of fracture*
0°–10°	75°
10°–15°	70°
15°–20°	65°
20°–30°	60°
30°–35°	55°
+ 35°	55°

(*b*) To the rise side,

75° at all seam inclinations.

(*c*) At the sides of the excavation on the line of full dip,

75° at all inclinations.

In overlying beds of flat formation,

(*a*) in marl 70°,

(*b*) in soft ground and quicksand 40°–50°.

No general value can be quoted for the angle of draw which, as a rule, is flatter than the angle of fracture by from 5 to 15 degrees, but it may be equal to the angle of fracture in exceptional cases. Every colliery should obtain their own observations by frequent and careful measurements in an attempt to obtain reliable values for both angles.

Section 3. Size of Extraction Area and Extent of Subsidence

It is extremely important to remember that the amount of the subsidence in the area affected is dependent upon the thickness, depth and dip of the seam, and the method of working adopted. A point on the surface is only subject to full subsidence if the worked-out area is of a certain size. All points on the surface are not affected to the same extent, the points near the edges of the affected area having the least subsidence and the maximum subsidence being recorded in the centre.

Three cases are considered in which the area of extraction and the extent of the subsidence area are compared:

1. Extraction of a partial area.
2. Extraction of a full area.
3. Extraction of a super area.

If the angles of draw plotted from the edge of the extraction area

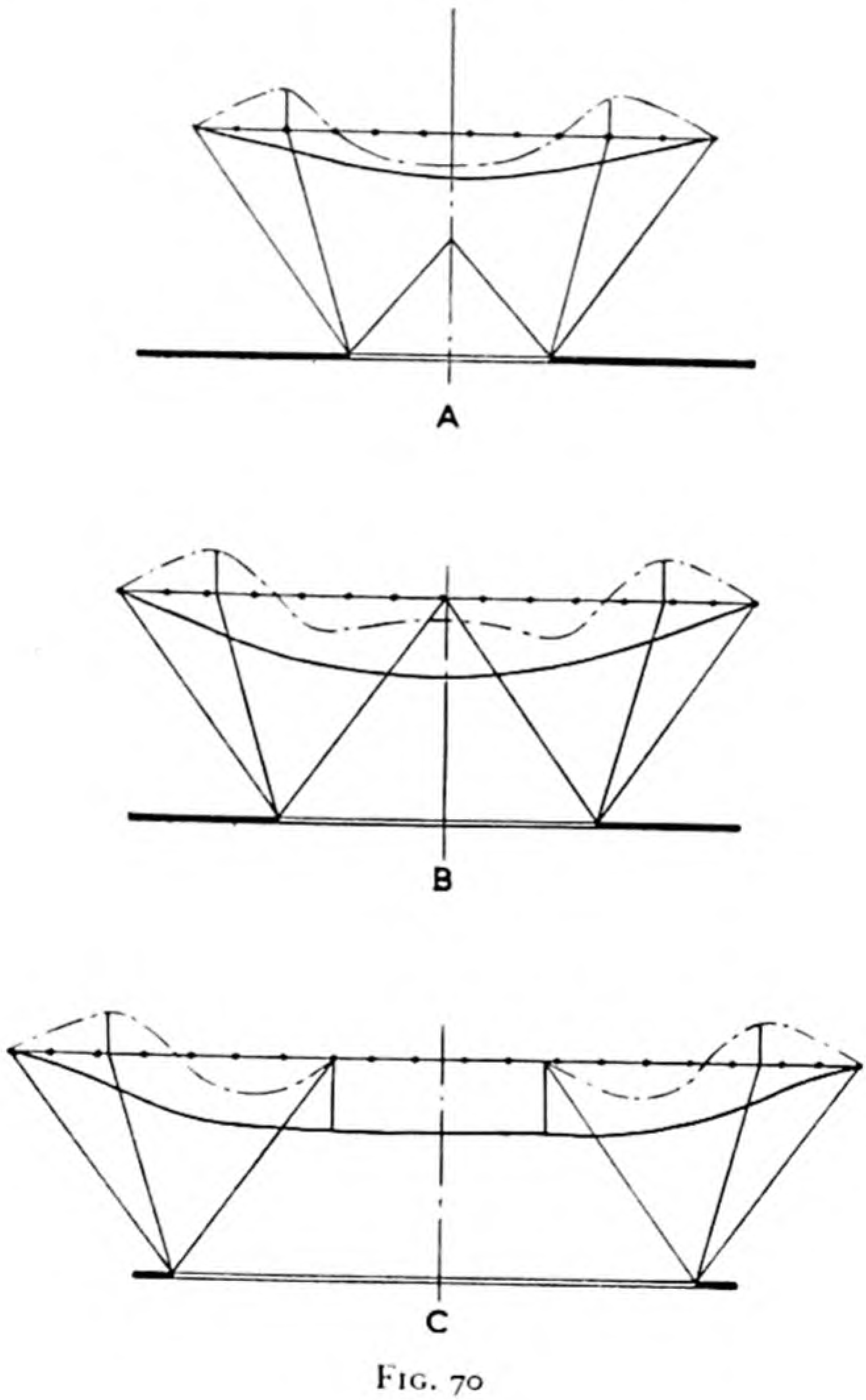


Fig. 70

towards the interior of the disturbed zone, on opposite sides, intersect below the surface, that is defined as a 'partial area', *vide* Fig. 70A.

The diagram in Fig. 70A represents the subsidence trough with an outer tension zone and inner compression zone. The conditions illustrated show that no point on the surface is subjected to the maximum

amount of subsidence possible, according to the thickness of seam extracted and the method of full extraction employed, solid packing, partial caving and full caving.

Maximum subsidence occurs only if the area is a 'full area', as shown in Fig. 70B, in which the draw, plotted from opposite sides of the area, reaches the surface. The surface at this point is subjected to the maximum subsidence, the subsidence trough is larger and, in particular, the compression in the centre is decreasing, indicating a return to normal. From here the subsidence is decreasing towards the sides of the affected area.

The subsidence conditions in a 'super' area, *vide* Fig. 70C, where the draw angle from opposite sides intersects the surface, are smoothed out considerably over the affected area. The tension and compression zones are, due to the return to normal above the centre of the extraction, limited to the edges of the affected area, which is extensive. In the centre of this area, the trough is flat and has been subjected to the maximum subsidence, but without any lateral differential movement.

It is obvious, when comparing these three cases, that the super area provides the most favourable conditions. The larger the extraction area, the closer these conditions are reached, in which there is no dangerous differential movement present due to a compressive wave above the centre of the extraction area. The horizontal tension and compression zones are limited to a small proportion of the affected area.

When considering the formation of these compression and tension zones above an advancing longwall face, it is clear that they will exist on all sides of the extraction area. The zones on each side of the face will advance with the face, but will only be fully developed when the coal face has stopped and the area of extraction worked out. The faster and more uniform the rate of advance in lower seams, the shorter is the time available for the development of such lateral zones and the better the conditions at the surface and in roadways, etc., in upper seams or horizons. It has been proved by experience that uniform and regular subsidence causes least damage to surface buildings; these are affected adversely by differential movement, due to horizontal tensile and compressive forces, by an oblique location in relation to the direction of advance. In order to reduce surface damage due to mining subsidence, working faces should have a high

and uniform rate of advance and the width of the working panel should be as long as possible, in order to make the area of excavation conform to super area conditions.

Section 4. Relationship between Subsidence and Depth

Since the three-dimensional zone affected by a working face is represented by an inverted truncated pyramid, it is clear that the volume and ultimate extent of the affected zone will increase with increasing depth. Fig. 71 illustrates the effect of depth on the affected zone, in the case of two seams worked in the same manner but at different depths, and also shows the effect of depth on the amount of

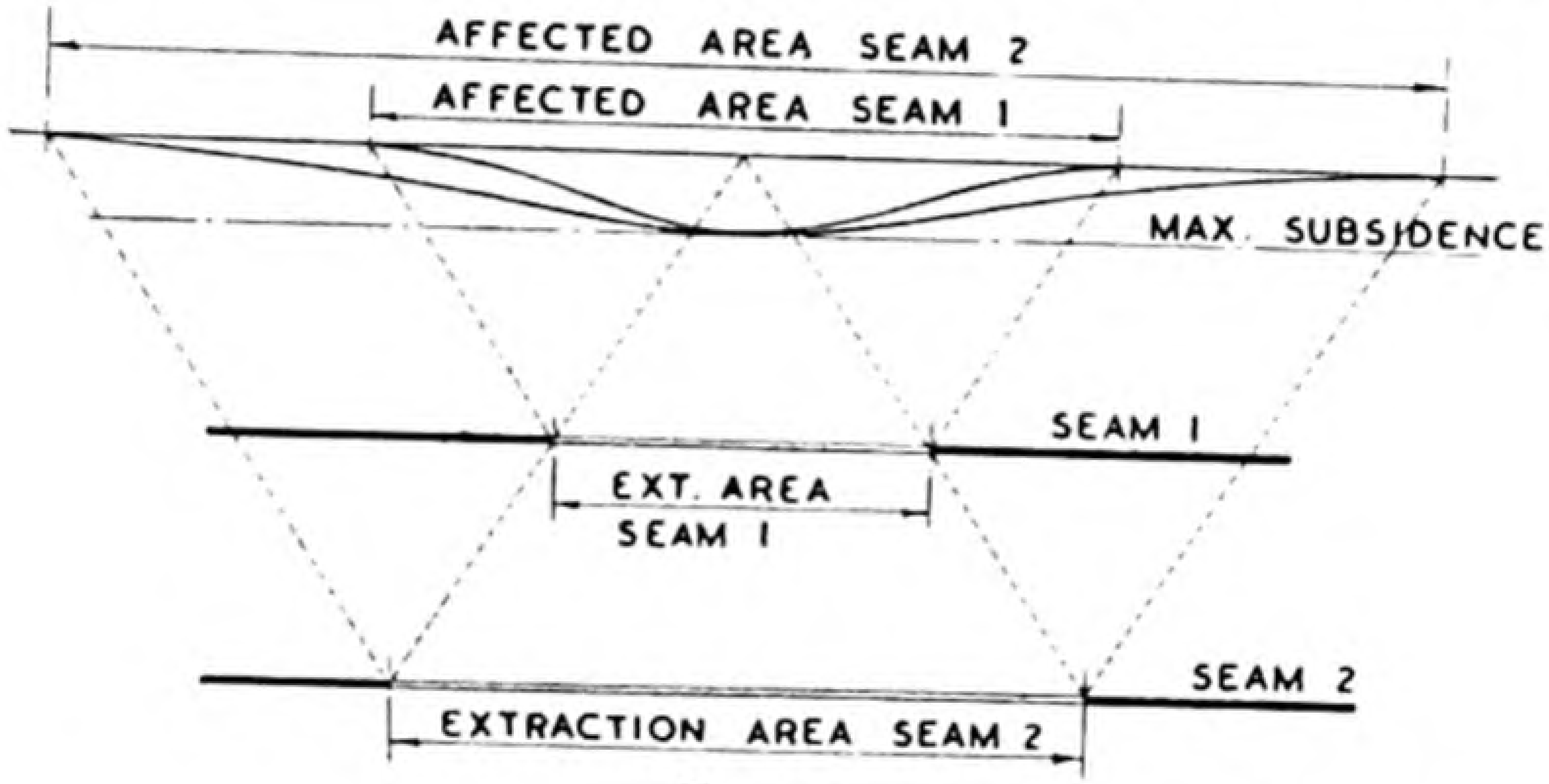


FIG. 71

subsidence. The extraction areas in seam 1 and seam 2 both correspond to 'full' areas in which the draw, plotted from opposite sides of the areas in each seam, reaches the surface. The subsidence trough is flatter in the case of seam 2 and the affected zone extends over a greater area. Only one point, at the intersection of the draw with the surface, is lowered to the maximum extent. An increase in the area of extraction in either seam would result in maximum subsidence being extended over a greater area, *vide* Fig. 72. From a comparison of these two cases it is clear that the 'full area' condition is affected by depth, the extraction area in the deeper seam being greater than that required in the lower seam. This is illustrated further in Figs. 72 and 73. For example, in the case of a seam at a depth of 490 yards, with an angle of draw of 60 degrees to the horizontal, the excavated area must have a diameter of 587 yards in order to reach full-area conditions, in which case only one single point on the surface is affected to the maximum vertical movement.

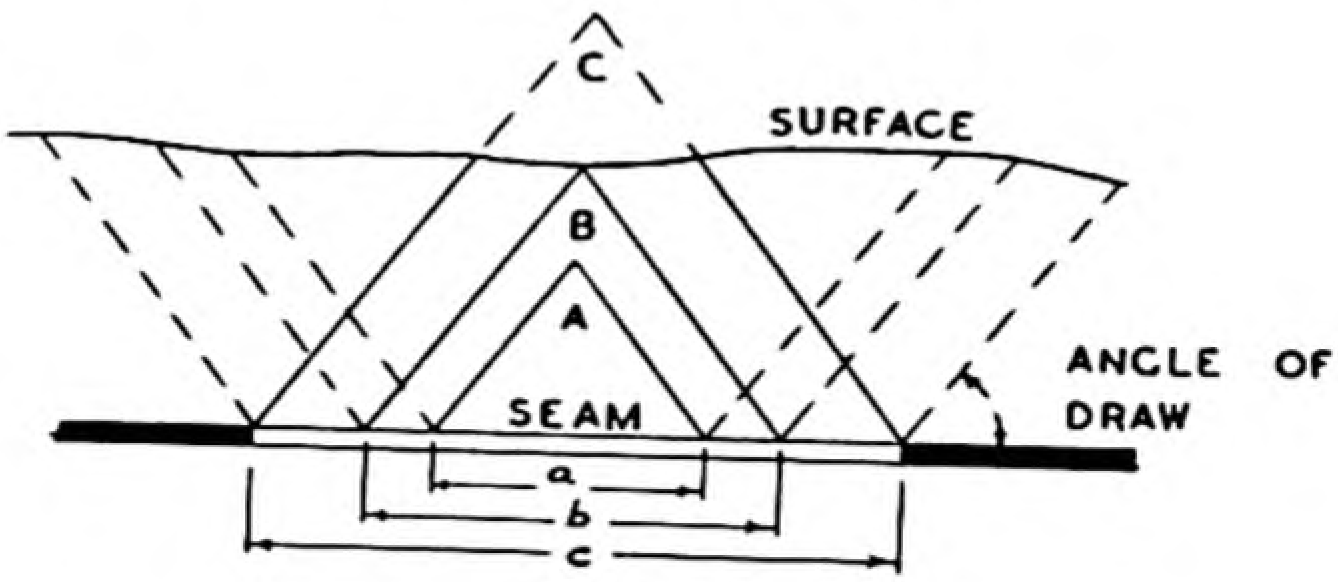


FIG. 72

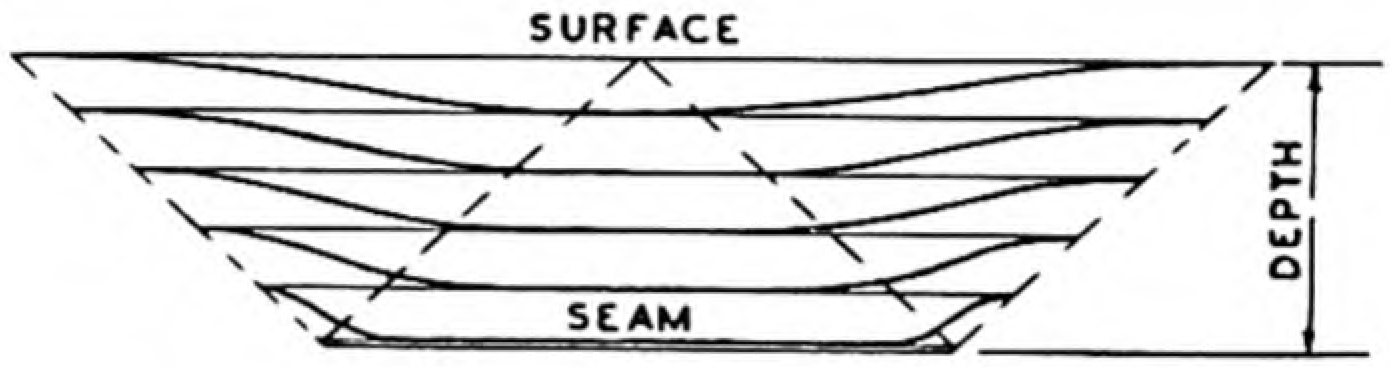


FIG. 73

This increase in the extent of the full area with depth has a major influence on any decision taken on the method of packing or caving to be introduced. The following table indicates the experience gained in the Ruhr on the degree of convergence of the roof, expressed as a percentage of the seam height of flat-seam working:

	Per cent.
Full caving	95
Partial caving	80–90
Dummy road packing	80
Hand-packing (normal)	60
Pneumatic stowage	40–50
Mechanical stowage (throwing belts, etc.) .	40–50
Hand-packing (carefully packed with well-built walls)	40–50
Hydraulic stowage	10–30

These average values are based on wide experience in the Ruhr-Aachen district and other mining areas on the Continent, although there may be variations in isolated cases. Three factors may influence the degree of subsidence of the roof with different methods of packing:

1. Premature roof lowering prior to packing.
2. The quality of packing material employed.
3. The efficiency of the pack building or stowage.

Premature lowering of the roof will be more likely with shale roofs than with sandstones. The extent of the initial convergence will be greater, the longer the lapse of time between extraction and packing. The pre-packing convergence expressed as a percentage of seam height will be greater in the case of a thin seam than in a thick one. Porous, moist, soft and slatey packing materials offer less resistance to consolidation than fine or coarse sandstone packing material. In the case of pneumatic and mechanical slinging stowage methods, the velocity of the stream of material is important, as well as the manner in which the material is directed into the waste. In the case of partial caving and dummy road packing, the width and quality of the pack strips will bear a great influence on the roof convergence.

With caving methods in, say, a seam 4 feet in height, the total lowering of the roof will be nearly 4 feet. In the same seam and adopting pneumatic stowage, the roof convergence would be reduced to about 2 feet. It would be wrong to conclude that the surface would subside to the same extent quoted in these two cases. The maximum degree of subsidence equivalent to the maximum roof convergence could occur only when the area of extraction is a full area and, in this case, only one point or a number of such points in line on the surface would be affected to the maximum extent. This maximum will occur only on a wider plane at the surface if the super-area condition is attained underground.

Consideration of these relationships indicates that, since the full area of extraction is enlarged with increase in depth, the total caving used at depth need not produce any greater surface subsidence than when any other method of packing is used in a shallower seam, provided that, in both cases, the seam thickness is identical and that a partial area is reached in the lower seam. If a super area is worked out in both seams, the amount of resulting subsidence at the surface will be in direct relation to the method of packing introduced. In the above example, in which caving is used in the deeper seam, the difference will be about 50 per cent. Another example can illustrate further this relationship. The case of two seams differing in thickness and depth but using the same method of packing is considered. The upper seam, 3 feet thick, is at a depth of 300 yards, and the lower seam, 6 feet thick, is at a depth of 600 yards. If a full area is extracted in the upper seam, the maximum subsidence will be 1 foot 6 inches. In the lower seam an equal area of extraction represents only a partial

area, and no point on the surface is affected by a vertical movement of more than 1 foot 6 inches, even if this seam is twice as thick as the upper seam and the same method of packing adopted. If a super area is worked in both seams, the maximum subsidence resulting from working in the lower seam will be twice as great as that from the upper seam extraction. If the area in both seams is of the same extent, then the surface area affected to the maximum degree will be less in the case of the lower seam. With the greater seam thickness at depth, the resulting horizontal tensile and compressive forces will be more pronounced than those occurring from working the thinner seam.

Section 5. Duration of Subsidence Activity

(*a*) *General introduction.* In the past it has been believed that subsidence over a working area, and within the disturbed zone, went on very slowly and over an extended period of years which may have been anything up to a decade or more. Under normal circumstances, subsidence is now considered to start slowly, quickly increasing in extent and remaining in this state for some time, after which the subsidence decreases and surface conditions are again stable. The length of the period of subsidence depends upon a variety of factors. The depth and nature of the strata are important as well as the condition of the strata, which may have been previously disturbed by the earlier working of another seam or seams. The extraction of seams at shallow depths will result in subsidence occurring earlier than in the case of deeper working. When working in a virgin area, subsidence will occur over a longer period than will be generally the case where previous working has taken place. In most cases, the ground movements cease after three to five years and, in some instances, movements have stopped one year after extraction has been carried out. It can be considered that 90–95 per cent. of the movement occurs in a period up to from three to five years after seam extraction, and the remaining small amount may extend over a further period, but may be hastened, if an additional seam is being worked. Fig. 74 is an example from a British coal field.

(*b*) *Changes of stress in the vertical direction.* It has been explained previously that strata movements caused by extraction will affect the stress distribution both in a horizontal and a vertical direction. The horizontal stresses have been described as occurring in tension and compression zones. The vertical stress distribution will result in a

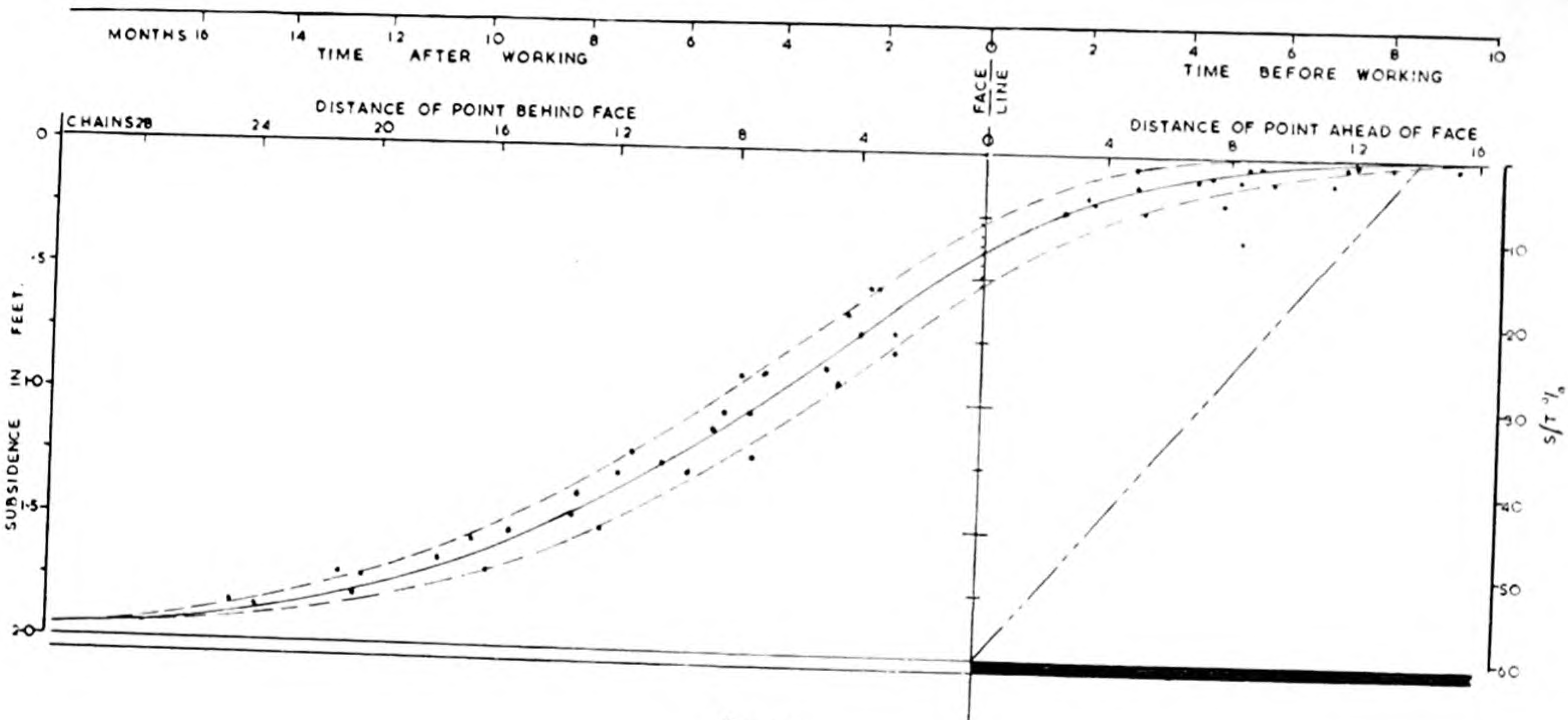

MONTHS 16 14 12 10 8 6 4 2 0 2 4 6 8 10
TIME AFTER WORKING
TIME BEFORE WORKING
FACE
LINE
DISTANCE OF POINT BEHIND FACE
DISTANCE OF POINT AHEAD OF FACE
CHAINS 28 24 20 16 12 8 4 0 4 8 12 16
SUBSIDENCE IN FEET.
0 ·5 1·0 1·5 2·0
S/T %
10 20 30 40 50 60

FIG. 74

condition in which there are relaxation zones and zones of high stress concentration.

The extent of these zones can be discussed with reference to Figs. 75A and B. These show the distribution of vertical stress to be added to or subtracted from the normal cover load to find the resulting vertical stress on a point in the affected area above a full and a partial area of extraction. The affected area, beyond which no strata movement will be caused by the extraction, forms an inverted truncated pyramid and is enclosed by the angle of draw, plotted from opposite sides of the excavation. The strata within this zone will subside at a rate which is zero at the boundaries and increases towards the centre of the affected zone. The differences in subsidence result in an increase in compressive stress within the area bounded by the angle of draw and by a more or less vertical line above the edges of the excavation. The amount of this additional vertical stress above a full area of extraction is shown at five levels in Fig. 75A. The additional compressive stress is increasing from the surface to the level of the excavation; it is zero at the drawline and on the vertical line above the edges of the excavation (A–A′ and B–B′). It reaches its maximum at the angle of fracture (A–A″ and B–B″). The adjacent zone towards the centre of the disturbed area is one of 'relaxed pressure' or a 'relaxation zone'. Above a full area the transition from the zone of high stress concentration to this relaxation zone will be on the straight vertical line above the edges of the excavation. Above a partial area, however, this changing from additional to reduced vertical stress will take place on a curved line (A–A″ and B–B″ in Fig. 75B). These zones of transition are called 'neutral zones'. Towards the centre of excavation the relaxation zone is bounded by the zone of 'full subsidence', which has the form of a pyramid, reaching the surface at point P in the case of the full area (Fig. 75A) and extending only to a horizon between levels 1 and 2 above a partial area (Fig. 75B). Within this zone of full subsidence the amount of vertical stress is normal in accordance with the load of the overlying strata. Above a full area of extraction the amount of relaxation within the zone of reduced vertical stress is symmetrical to the additional stress distribution in the compression zone. It also increases with depth and is zero at the neutral zone above the edges of the extraction and at the boundary of the zone of full subsidence. Considering a partial area of extraction, the relaxation zone on the

upper levels is not interrupted by the zone of full subsidence, which does not extend to these levels. On these higher levels the maximum relaxation, which increases with depth, is found above the centre of the extracted area.

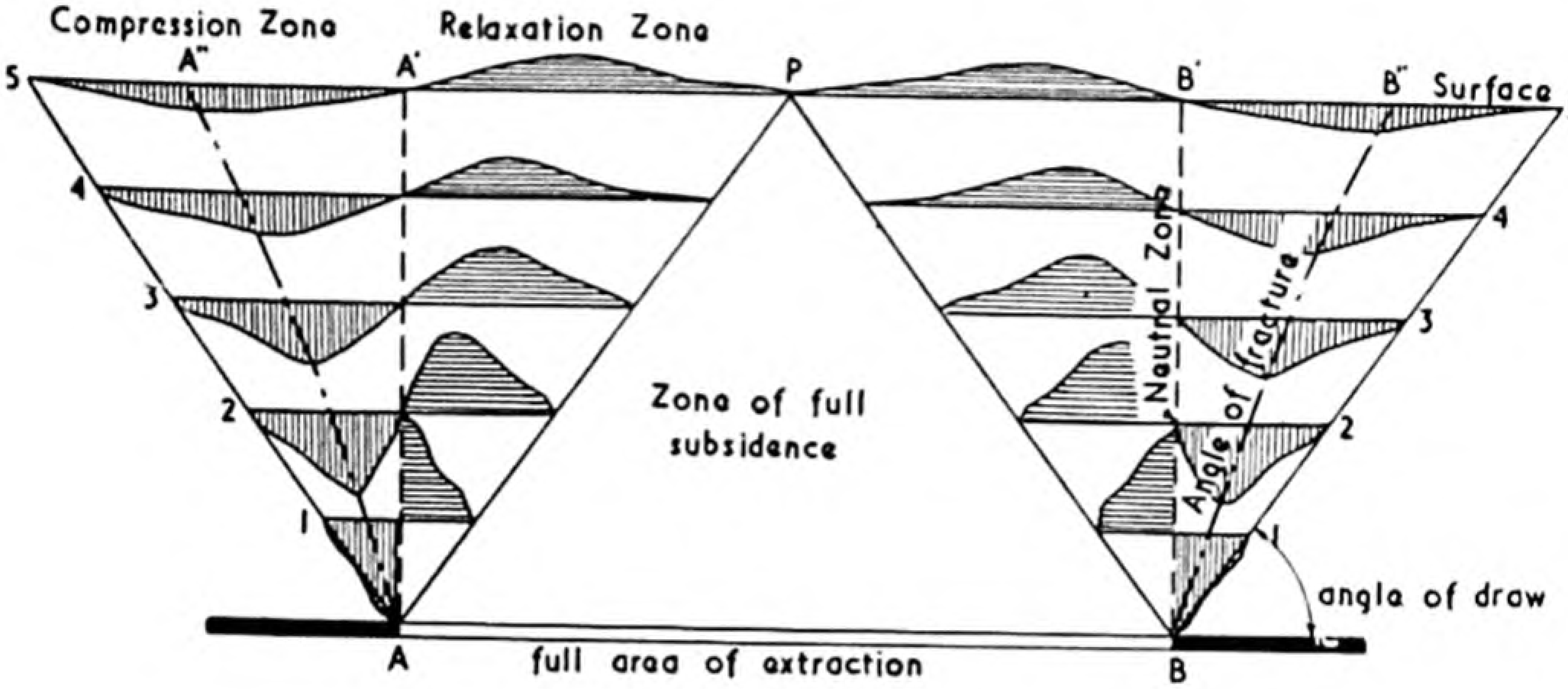


FIG. 75A

(WITH ACKNOWLEDGMENT TO PROF. O. NIEMCZYK.)

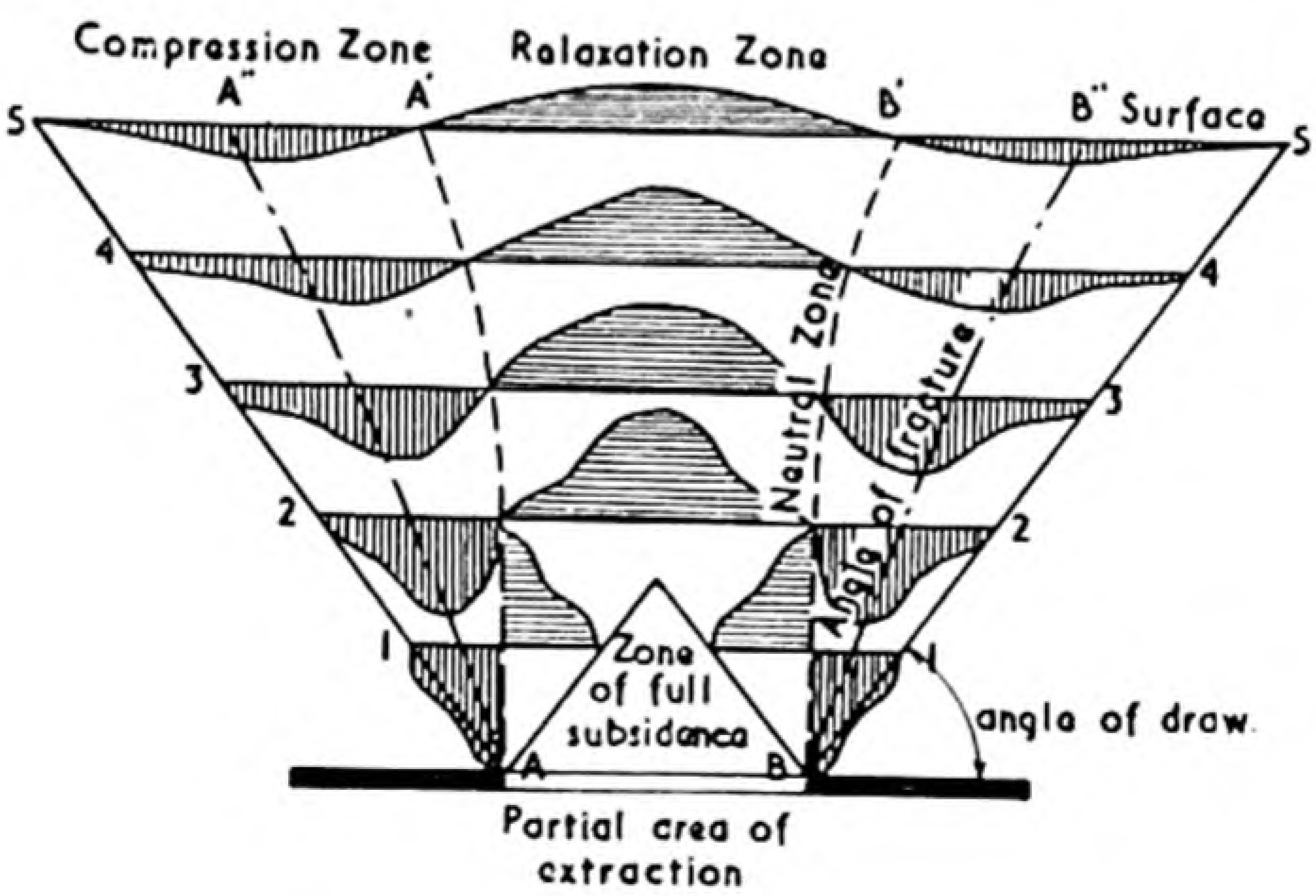


FIG 75B

(WITH ACKNOWLEDGMENT TO PROF. O. NIEMCZYK.)

PART II

DAMAGES DUE TO MINING OPERATIONS

Section 1. Surface Damage

The strata movements caused by underground mining operations may affect building structures, railways, watercourses, etc., as well as cause changes in surface level, affecting drainage and sewage communications. The changes at the surface may appear in many variations: simple vertical lowering of the surface level; lowering of the surface level by sideways movement, thus tilting buildings and creating fractures in the structure and producing various combinations of tensile and compressive stresses on the surface, causing a wide variety of differential movements. Where mining is being carried out below an overlying brittle bed, such as a sandstone or limestone in which fissures may exist, the sudden bursting of these fissures may cause a very minor earthquake. In proportion to the other consequential damage, the probability of such occurrences is small. Simple uniform subsidence, even when accompanied by small variations in tensile and compressive surface stresses, create little surface damage, especially if the base area is small. Larger buildings can also withstand such minor subsidence if it is uniform. The larger the building or structure, the less likely is the chance of uniform subsidence and the greater the possible damage or interference. The effect of even minor subsidence on watercourses with a low rate of fall may cause flooding and the ultimate destruction of the land for full utilisation. Canals, roads and railways can be affected in a similar way unless special precautions are taken.

In surface areas where tensional stress is produced, the typical result is steeply inclined fractures of building walls, cracking and stepping of the ground surface, extension of railway track joints and the breaking of water, gas and electricity supply services. On the other hand, in areas where compressive stresses are produced, the surface movement is marked by the uplifting of kerbstones, flat fractures in buildings and walls, the bending of railway tracks and the fracture of underground mains. Surface installations, such as tramways, which are more firmly attached to the ground will suffer more from lateral movement, and the damage is likely to be greater to tramway than

railway tracks. Positions on the perimeter of subsidence troughs are not suitable for buildings of any kind.

Section 2. Measures Adopted for the Reduction and Prevention of Damage to Surface Structures

It is apparent from an investigation of the Ruhr district of Germany that there are methods which can be adopted in order to prevent or reduce the effects of mining subsidence. The Ruhr, with its collieries, blast furnaces and factories, is densely populated and traversed by many roads and canals. Large areas in the Ruhr district have already been lowered by 5, 10 and even 15 yards, and will sink even farther. The cost involved in applying these measures increases the total cost per ton, on the average, by from 2 to 3 per cent., and in some instances the increases may be from 5 to 10 per cent. during certain years.

The measures adopted underground for the full extraction of the coal are specially important and are discussed in detail in Chapter 2, Part III, Section 3 (*b*), but brief details of the factors involved in building and works construction at the surface can be discussed and the possibilities indicated. Long extended lines of houses, 'tied' to each other, and large openings or niches, especially in supporting walls, should be avoided. Important buildings should be erected on proper reinforced raft foundations. In the design of embankments, sufficient formation width should be allowed so that, in the event of settlement of the surface, the formation level can be regraded to the original level; bridges should be designed to enable them to be raised without the necessity for further reinforcement; railway tracks should be laid with expanding joints capable of withstanding lateral movement resulting from tensile and compressive forces. Similarly, with underground pipe-lines, stuffing boxes should be used to give the line a degree of flexibility, and buried cables should be provided with an expanding joint and the cable should be coiled in sections to allow for lateral and vertical surface movements. In subsiding areas, foundations, walls, etc., should be built with a gap construction to absorb differential ground movement. The three-point roof-strut construction has been proved desirable, since this design retains its balance even when the roof slants due to ground movement. Particular care should be taken in the grading of watercourses to allow for possible changes in the surface level which may hinder flow, so that it is usual

to allow a steeper initial gradient before underground extraction takes place.

Section 3. Pseudo Damages

It is usual to find that in mining areas any damage to surface structures is attributed to underground working. This is not always the case and it should be understood that the terms subsidence and surface damage are not synonymous. There are many instances of buildings and other surface structures, remote from mining areas, showing damage which cannot be distinguished from that found in mining districts. The damage may be due to defective building construction or materials, or poor foundation construction. The recurring shocks from heavy traffic may be another reason for such damage. Where the surface subsoil has a poor bearing strength, such as the mud and quicksand in some London areas or the presence of peat in the Berlin area, the effects of poor foundation construction can be disastrous. Building near to outcrop areas, in which the strata are inclined to slip, could also be a cause of damage to such structures. Tramway track, badly laid in poor subsoil, may be affected in the same way and further aggravated by poor maintenance on roads where the other traffic is heavy.

In the Doncaster area, the lowering of the general water-level has resulted in damage to some surface structures, forming typical examples of pseudo damage, although in the same area the surface damage resulting from subsidence due to mining is considerable.

Section 4. Underground Subsidence Effects

It is usual to consider subsidence and its effects only with reference to the surface, but the same effects can be considered with reference to underground installations, such as drifts, staple and main shafts with all their ancillary supply installations. Damage to underground constructions or drivages will also occur in regions within the movement zones set up near winning faces, as previously described. In the same way as on the surface, differential strata movement will cause damage, resulting in the bursting of roadway linings, changes in the gradient of roadways, deviation of shafts from the original vertical position and damage to fixed machinery foundations. Damage underground will not be confined to the internal zone fixed by the internal draw angle, or pressure arch, but in the adjacent lateral areas near the excavation and below. The lining and support of roadways, there-

fore, has to be designed to withstand this movement and counteract probable damage, and precautions have to be taken to minimise both surface damage and the effects of the strata movements on underground installations.

PART III

METHODS OF WORKING DESIGNED TO REDUCE OR PREVENT SURFACE DAMAGE AND AT THE SAME TIME PROVIDE SUITABLE STRATA CONTROL UNDERGROUND

Section 1. *General Measures*

It has been shown in Part I, Section 3, that the movements of the affected area at the surface, resulting from working a partial area, are unfavourable, since high tension and compression zones are introduced over the whole area with abrupt changes between these extremes, resulting in differential surface movement likely to cause severe damage to surface structures.

Similarly, it has been shown that the super area of extraction provides an easier means of control and elimination of the differential surface movements. In this case, the tension and compression zones are confined to the edges of the surface area and, immediately above the working area, the surface subsidence is uniform in a vertical direction. Many surface structures can be designed to withstand uniform vertical movement, whereas differential transverse and vertical movement is difficult and costly to control. From all these considerations it has been concluded that, with regard to the mitigation or elimination of damage to surface structures, short faces are to be avoided and long faces preferred. The working length of the face in the direction of advance should be chosen so as to reduce the number of rib-sides, or boundaries, where tension and compression zones will be set up at the surface, that is, areas where differential lateral movement is introduced. For the same reason temporary interruptions in the face advance are to be avoided, otherwise the rib-side condition is produced. Rapid face advance is important in order to reduce the amount and effect of lateral movement. It is also better to develop diverging faces from a single cross-cut extending to the neighbouring cross-cuts than to have converging units between cross-cuts. In many instances this cannot be carried out for other mining reasons discussed in Chapter 1, Part II, Section 2 (*d*).

An advantage of diverging faces is that it is better to work away from the cross-cuts in order to reduce maintenance costs on the levels. In the case of diverging faces developed away from a single cross-cut extending to the neighbouring cross-cuts, these are affected by the forward-moving front abutment zones. With the method of development of diverging faces from every cross-cut, the strata half-way between cross-cuts are subjected to opposed front abutment zones, with resulting opposite lateral movement at the surface and underground, and the area of extraction may be limited to less than a super-area condition depending on the depth of cover because of the introduction of this central zone. With regard to the elimination of too many rib-sides, it would be better to proceed with diverging face development from both sides of the cross-cut simultaneously extending to the neighbouring cross-cuts than to finish working between one pair of cross-cuts before starting on the other face. In the case of the diverging faces, only one large area of activity is produced, whereas two smaller areas are introduced by converging faces between cross-cuts. In the latter case, it becomes more difficult to provide super-area conditions with increasing depth. It has been shown in Chapter 2, Section 5, that to provide the favourable super-area condition at a depth of 500 yards requires an extraction area too large to be mined with one face and also a similar parallel unit adjacent to the face. Fig. 76 illustrates such a case, in which a group of three seams is to be developed, using five staple shafts. If the three seams were worked successively in depth one after each other, between staple shafts 1–2, 2–3, etc., four pronounced subsidence troughs would be formed with their boundary tension and compression zones formed in succession. If the development of the group of seams were continued, simultaneously, in the four staple-shaft districts, by advancing at the same time with four faces in seam 1, then super-area conditions could be attained. In Fig. 76 the flat subsidence troughs are shown following the development in each seam in depth. The same principle can be applied to any sequence of working the three seams in depth, required by market conditions or coal quality, in the four staple-shaft districts, provided that each staple-shaft district continues in operation.

Finally, the dangerous condition which will arise if the faces are stopped along the same vertical line, as would happen if the seams were worked to the same boundary, can be avoided by staggering the

finishing position of the faces in each seam in depth; the effect will be increased the nearer the seams are together and the greater their thickness. Fig. 77A shows the fracture at the surface resulting from the working of two and three seams in close proximity. If the workings were advanced to the limits shown in Fig. 77B, the possibility of a

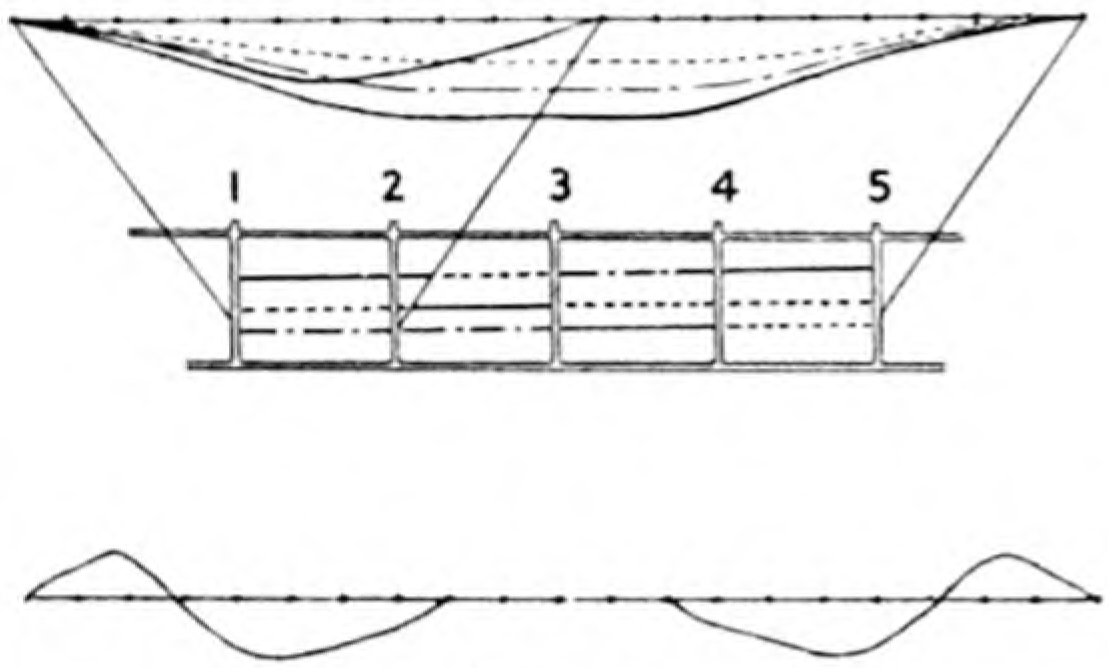


Fig. 76

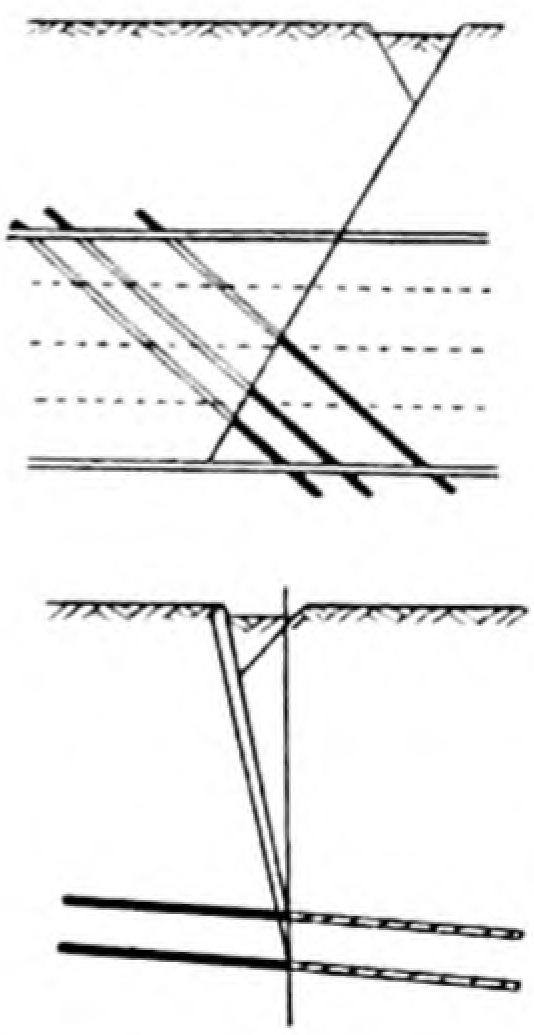


Fig. 77A

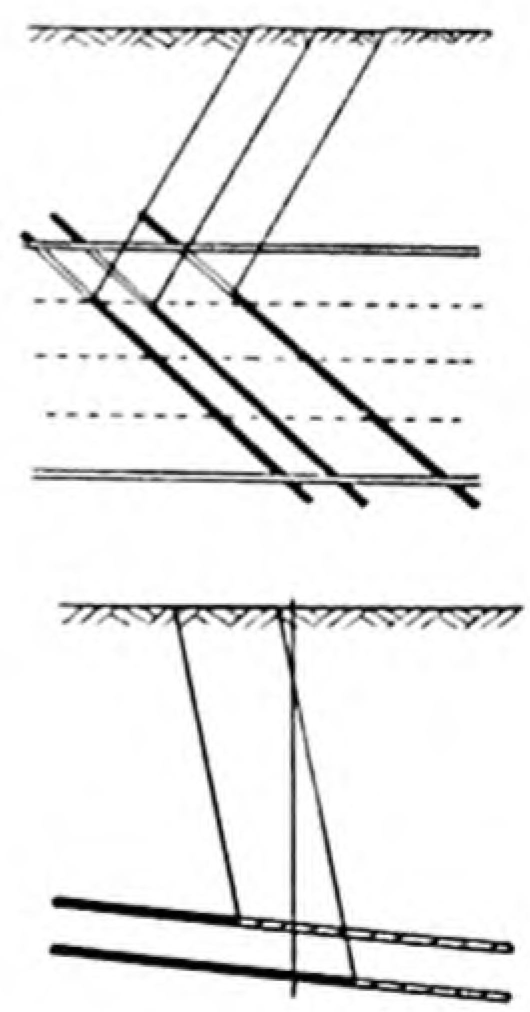


Fig. 77B

surface fracture is diminished and the boundary area of the subsidence trough is not concentrated along a narrow zone.

Section 2. Safety or Protection Pillars

(*a*) *Definition*. Safety pillars are pillars left for the protection of the surface and underground installations, and are designed to obviate or minimise strata movement caused by mine working.

(*b*) *Protection pillars for the surface.* Important surface buildings which have to be protected may include the mine installations, factories, gas works, power stations, railways, public institutions (such as hospitals and churches), canals and other waterways. The dimensions of the pillar will depend on its depth and on the surface area to be protected, the dimensions being greater with increase in depth. Thus, accurate information on the size of the external angle of draw is important, since the extent of the reserves sterilised by the protective pillar will depend on the depth and draw to the surface. Protective pillars in which the draw has been limited to the vertical are to be avoided, since the extent of the affected area on the surface is governed by the depth and the draw angle for the prevailing conditions, according to the theory described.

The decision to leave protective pillars must rest on the extent of the public interest and on economic considerations. The inevitable loss resulting from sterilisation in one or more seams will affect both the profits and life of the mine. A decision can be reached only by expressing a balance between these factors and the probable compensation for damage, and additional mining cost necessary to provide appropriate safeguards in winning and stowage which would minimise the surface damage and the extent of liability for compensation.

(*c*) *Protection pillars underground.* The use of protective pillars underground may be confined to boundary barriers, shaft pillars and pillars for the protection of main roads. Boundary barriers have been introduced as a protection from workings which may become flooded in adjacent collieries and, in some instances, have been left to maintain ownership limitations. They have also been provided to separate the ventilation systems of one colliery from another. Since the adjacent collieries both leave a barrier, the loss in reserves is divided equally.

The main purpose of the shaft pillars is to provide protection for the shaft and immediate surface buildings from adjacent mine workings. The size of the pillar is calculated on the same principles as for a surface protection pillar.

Roadway protection pillars may be left for the protection of main roads, their size being dependent upon the prevailing conditions.

(*d*) *Application of protection pillars.* The great disadvantages of coal pillars left for support, especially for shafts and surface buildings,

together with the advance in the knowledge of strata movements in mining areas and surface subsidence, have shown the necessity that such pillars should be avoided as far as possible. On the Continent the view is expressed that shaft pillars are justified only if the coal-measure strata is overlain by running sands or is heavily water-bearing. In such cases, the working of the coal within the shaft pillar would increase the danger of an inrush of water or sand into the shaft, where the shaft tubbing is affected by strata movement. This danger must be avoided at any cost by leaving the shaft pillar for protection. Where such circumstances cannot arise, the practice is to work the shaft-pillar area, taking into consideration all possible precautionary measures to minimise damage to the shaft lining. In the case of staple shafts, protection pillars should not be required. The question of staple shafts is discussed in Section 3, and it will be seen that it is possible to work round the shaft and still maintain it.

The importance of boundary pillars and protective barriers has decreased with the unification of royalties and the enlargement of colliery units by the combination of several smaller collieries.

Coal pillars left to protect main roads driven in stone can prove to be a disadvantage; further details of such cases being given in Section 4. In the case of recent developments at a depth of 1,700 feet, where a yielding pillar technique has been employed for roadways driven in the seam, the main roadway pillar is designed to enhance the control of the upper strata beds by the transference of the main roof load to side abutment areas, either in the solid coal or in the goaf. This pillar is left during the initial development, but taken out on completion of the main panel extraction. It is often found that leaving pillars under rivers and canals has now ceased, the trend being to develop and conserve the available resources. Where working is being conducted under canals, it has been the experience on the Continent that an even subsidence can only be obtained by changing the packing method employed in each section, where the number and thickness of the workable seams varies. If all the seams were worked and the same stowage system employed in every case, unequal subsidence results and the flow of the canal system is impaired. Mechanical stowage is carried out where the seam density is large and, in some cases, one or two seams may be sterilised, while, in regions of low seam density, partial or full caving methods are employed.

Section 3. The Working of Shaft Pillars

(*a*) *The effect of shaft-pillar extraction.* The strata movement above a worked-out area has already been explained in Part I, Section 1. The strata is affected by induced tensional and compressive forces due to lateral and vertical movement above the excavation. The effect of this movement at the surface is either a gradual vertical subsidence over a wide area or a differential lateral and vertical subsidence. A shaft which is situated vertically within the zone of influence of a working area is exposed at all times to lateral as well as vertical movement, unless special precautions are taken to minimise the effect of these tensile and compressive forces. The principle employed to attain this aim is to produce tensional and compressive effects in the same zone at the same time. This is done by working faces simultaneously within the shaft pillar, the faces being distributed in such a manner that the tensional effects produced by one face are counter-balanced by the compressive effects produced by another or vice versa. If a complete neutralisation is not effected, at least the unfavourable conditions are minimised.

The system of organisation and distribution of working faces designed to produce this effect has been called, by K. Lehmann, the 'Harmonic Method of Working'.

The most favourable position for the shaft in relation to the winning faces can be described with reference to Fig. 78. If the shaft is situated at point 8, between the boundary and the centre of the area to be worked, the lower part of the shaft is within a zone affected by vertical movement only. The upper part of the shaft lies within a zone affected by tensile and compressive lateral forces resulting in both lateral and vertical movement. From Fig. 78 it can be seen that the nearer the shaft is to the centre of the working area, the less is the effect of the lateral movement. If the shaft position coincides with the centre of the area, the unfavourable condition of the lateral movement zone in which stress is concentrated is obviated, the shaft being situated within the zone in which only vertical movement without any tensional or compressive effects occurs. This will only be the case, however, where the area being worked corresponds to at least a full area.

It has been concluded from experience on the Continent, especially in Germany and Holland, that the best position for a shaft around

which the pillar is to be extracted is in the centre of a full area. In other words, a full area must be extracted in each seam intersected by a shaft situated in the centre of the full area, if the shaft is to suffer the least possible damage.

In view of the fact that the extraction of a full area requires time, the situation of the shaft within the area is not the only condition to be considered. The faces must be arranged within the area in such a manner and worked in such a sequence and direction of advance that,

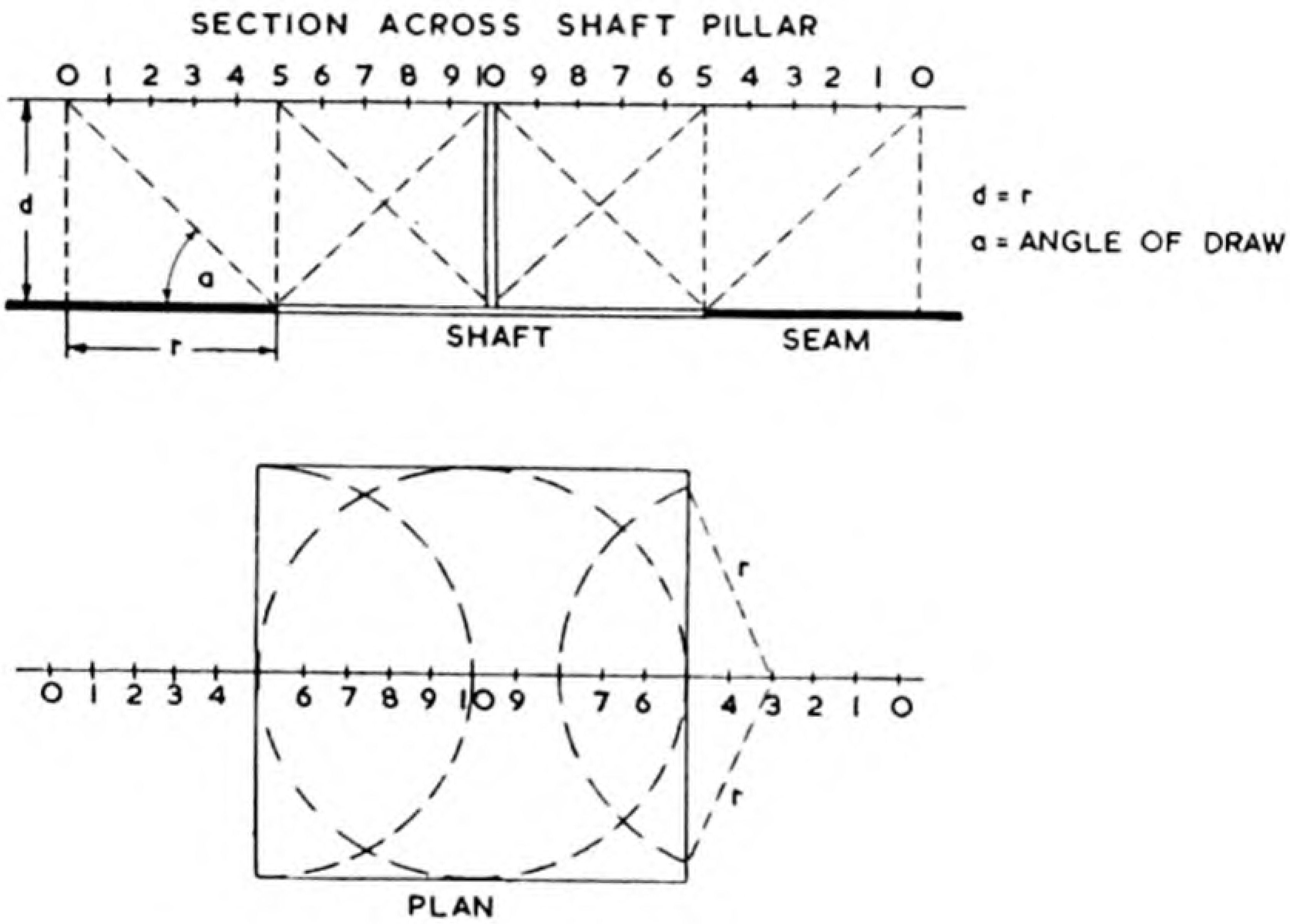


FIG. 78

during the extraction of the full area, the tensional and compressive effects and lateral movement are minimised, i.e. the Harmonic Method of working must be applied.

For example, if the faces start from the centre of the area and advance to the boundary, the lower section of the shaft subsides vertically without being influenced by other lateral effects.

Where the shaft passes through the seam, the immediate roof subsides 100 per cent., and the shaft lining must be removed to allow for this subsidence. The subsidence decreases higher up the shaft beyond the point where the area worked is a full area for that particular depth. In consequence, the shaft will be in tension after this

point of full subsidence, the tension decreasing towards the surface. The vertical movement of the shaft will cease when a full area corresponding to the total depth of the shaft has been worked.

Should the faces advance from the boundaries of the shaft pillar towards the shaft, the upper section of the shaft suffers the first subsidence effects, and the lower portion is not affected. In this case the vertical compression effects will be serious. As the faces continue to advance towards the shaft, the compression zone extends in depth until it reaches the shaft and seam contact, when the shaft movement ceases. But the upper section of the shaft suffers the effects of compression first, then tension and finally compression as the faces near the shaft and the centre of the area. These effects are first observed in the upper section, then in the middle section and finally in the lower part of the shaft.

The effect of face advance, from or towards the shaft, on the lateral movement of the strata is also important. The faces can be distributed within the area so that the horizontal movement is neutralised but, if only one part of the area is being worked, the shaft will tend to incline towards the worked-out area. This condition can be avoided only by introducing a comparable working face on the opposite side of the shaft. It is obvious that such alternating movements taking place at different periods would cause damage to the shaft. In order to prevent this arising, it is necessary to organise the face development so that this is done simultaneously. The horizontal or lateral movements are neutralised and the shaft remains vertical. It is also possible to compensate or reduce the effect of vertical tension and compression. The following example of the conduct and organisation of simultaneous working within a shaft-pillar area will illustrate the technique employed.

(*b*) *The extraction of shaft pillars by the compensation-stress technique.* If a full area within a shaft pillar is to be extracted, using the stress-compensation technique, careful observation and calculation of the extent and progress of subsidence and other movements is necessary.

Fig. 78 shows a plan and section of a full area for a level seam with the shaft in the centre. Since only straight faces are possible, the full area to be worked is set out as a square and the inscribed circle to this square is drawn. The angle of draw is assumed to be 45 degrees. In order to calculate the resulting subsidence, the section through the

centre is subdivided into 20 parts at the surface, with 10 parts on each side of the shaft, numbered from left and right towards the shaft centre, point 10. Point 0 will not be affected, being on the limit of the affected area, but point 1 is affected by that part of the seam extraction vertically below the section 5–6; similarly, point 2 by the extraction below section 5–7, point 3 by section 5–8, and point 5 by section 5–10.

Point 10 will be affected by the extraction of the complete area, that is, vertically between point 5 on either side of the shaft. The points are subjected to varying degrees of subsidence due to their different positions and the relevant areas of extraction affecting them. Point 10 suffers 100 per cent. of the maximum subsidence, whereas

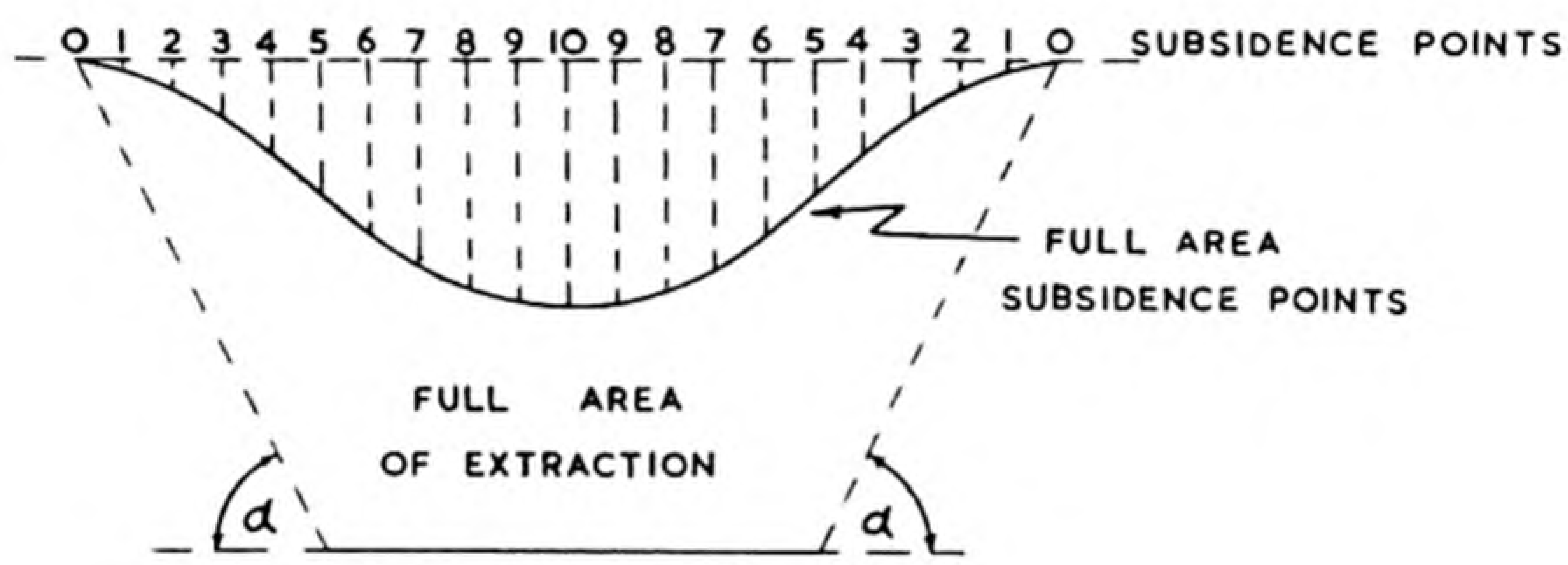


Fig. 79

point 5, which is only affected by half the area of extraction, suffers 50 per cent.; points 1–4 are affected by less than 50 per cent., and points 6–9 by more than 50 per cent. In the cross-section shown in Fig. 79, the percentage of maximum subsidence is plotted against each point position from 0–10 to the left and right of the shaft centre. Utilising these figures, the subsidence graph can be drawn, as shown in Fig. 80. This is a normal or standard curve from which the probable subsidence can be calculated if the areas of extraction are regular in dimension and the seams are level.

If, for example, the extraction of the full area is not completed and only that part under section 5–8 has been extracted, points 1–3 are subjected to the normal regular subsidence of 2 per cent., 8 per cent. and 18 per cent. of the maximum subsidence respectively. In the case of point 4, however, since it is not affected by the extraction of the section 8–9, the subsidence will be less than normal and the effective

subsidence is obviously the same as for point 1, when section 5–6 is extracted. In this case, therefore, the probable subsidence at point 4 can be found by subtracting the subsidence at point 1 from the normal maximum subsidence at point 4. The probable subsidence at point 5 can be found in the same manner, since it is not affected by the extraction of the section 8–10, and the extent of the subsidence corresponds to the normal maximum subsidence at point 2.

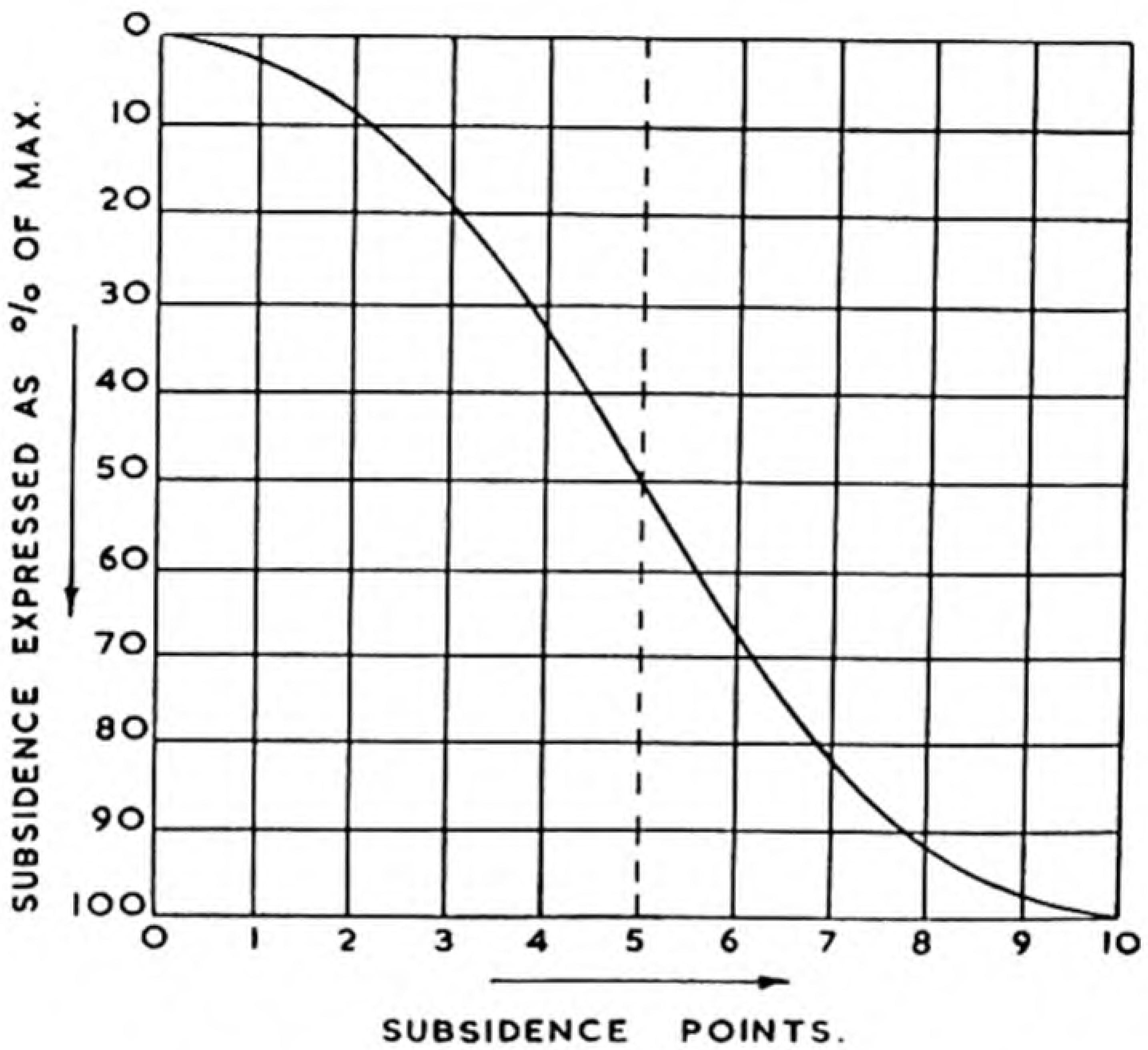


FIG. 80

In the case of the extraction of section 5–7, point 5 is affected, and the normal maximum subsidence at point 3 is subtracted from the normal subsidence at point 5 to arrive at the probable subsidence at that point.

If extraction extends over the 4 sections, between points 5–9 (left side), points 1–4 will be subjected to the normal maximum subsidence, while the remaining points, 5–9, will be affected to the following degrees of subsidence:

Point 5. Normal subsidence point 5 minus normal subsidence point 1.

Point 6. Normal subsidence point 6 minus normal subsidence point 2.

Point 7. Normal subsidence point 7 minus normal subsidence point 3.

Point 8. Normal subsidence point 8 minus normal subsidence point 4.

Point 9 (left). Normal subsidence point 9 minus normal subsidence point 5.

Point 10. Normal subsidence point 4.

Point 9 (right). Normal subsidence point 3.

Point 8. Normal subsidence point 2.

Point 7. Normal subsidence point 1.

Since the curve is symmetrical about point 10, it is sufficient in this case to work out the rates of subsidence on one side of point 10 only. In the two examples given, the extent of the worked-out area is given in zonal sections, the widths of which correspond to the distance between two successive points along the section line in Fig. 78. The actual width of the section is dependent upon the depth to the seam and the angle of the draw.

With an assumed angle of draw of 45 degrees, the width of the zone-section is one-fifth the depth, since the line of draw is the hypotenuse of a right-angled isosceles triangle, the sides of which are equal to the depth. With a depth of 400 yards, therefore, the zone-section is $400 \div 5$, or 80 yards. If the worked-out area in the seam is 300 yards wide at that depth, the area would be equal to $300 \div 80$, or 3·75 zone-sections. The subsidence curve for that area is obtained by marking out the subsidence rates at points 1, 2, 3 and 3·75, while for points 4, 5, 6 and 7 the difference in the rate has to be calculated between these points and those which are included in a length of 3·75 zone-sections to the left; that is, the points 0·25, 1·25, 2·25 and 3·25. The point 7·25 corresponds to the zero point 0 on the other side of the working area and is not affected.

In a similar manner, the subsidence affecting a shaft, due to the shaft-pillar extraction, can be calculated. Fig. 81 shows a shaft section in which the distance *AB* represents the right-side half of a full area in a level seam. It is assumed that the full area can be completely extracted in one year, and the seam is divided into 4 quarterly zones, *a*, *b*, *c* and *d* of 3 months' extraction, with an assumed angle of draw of 45 degrees. The section of the shaft affected by extraction of zone *a* extends to point 4 in the shaft, and zones *b*, *c* and *d* to points 3, 2

and 1 respectively. If the half full-area, represented by *AB* in Fig. 81, is sub-divided into 5 zone-sections, so that point 10 is vertically above *A*, and point 5 vertically above *B*, the point 4 is affected by one-quarter of the 5 zone-sections, point 3 by one-half of 5 zone-sections, point 2 by three-quarters of 5 zone-sections, and point 1 by all 5 zone-sections.

The influence of the extraction of area *a* on the four shaft points 1–4 may be considered first, area *a* being also divided into 5 zones of equal width. Point 1, being farthest away, is affected to the least extent, and point 4 is subjected to the effect of maximum subsidence. The maximum subsidence is equal to the effect of the 5 zone-sections. Point 3, mid-way down the shaft, is only affected to half the maximum extent of 5 zone-sections, point 2 by one-third of the maximum and point 1 by one-quarter the maximum extent. The same reasoning can be applied to area *b*, which has a width of 2 sub-sections (six months' working), and the resulting subsidence rates are as follows:

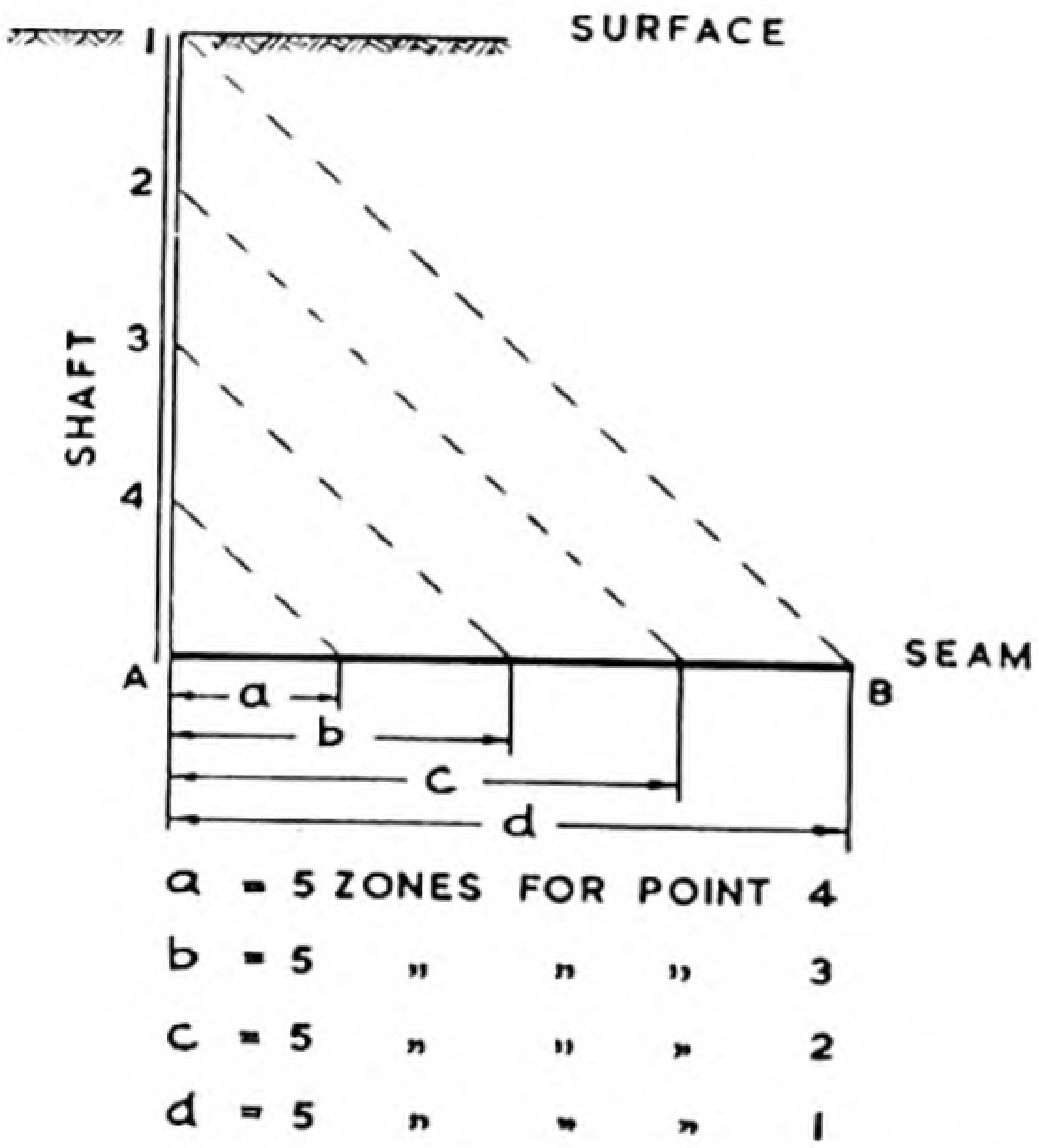


FIG. 81

Point 1 is affected by one-half maximum rate of 5 sections.
Point 2 is affected by two-thirds maximum rate of 5 sections.
Point 3 is affected by the maximum rate of 5 sections.
Point 4 is affected by the maximum rate of 5 sections.

Areas *c* and *d* can be treated similarly.

To calculate the subsidence of point 1 after the extraction of area *a*, reference is made to the subsidence curve in Fig. 80. Point 1 is affected to the extent of the extraction of 1·25 zone-sections. Thus, the normal maximum subsidence resulting from the extraction of area *a* (or one-quarter of 5 zone-sections) must be reduced by the subsidence of a point 1·25 to the left of point 5 on the graph. This

point is therefore 3·75 on the graph, at which point the subsidence is 28·5 per cent., and at point 5 is 50 per cent. Thus, the subsidence resulting from the extraction of area *a* affecting point **1** is 50—28·5, or 21·5 per cent. of the maximum subsidence.

The method of calculation is the same for point **2**, which is affected by one-third of the 5 zone-sections, or 1·67 effective zone-sections, in the first three months' working. The maximum subsidence at point 5 must be reduced by that at point 3·33, i.e. 50—22·5, giving 27·5 per cent. as the subsidence at point **2**. Similarly, the subsidence of points **3** and **4**, after extraction of sub-area *a*, are 36·8 and 50 per cent. of the maximum subsidence.

After six months' extraction (the end of the second quarter of the year) the area extracted has extended to *b*, which can again be divided into 5 zone-sections. In this instance, points **3** and **4** are affected by all zone-sections and suffer a subsidence equivalent to point 5 on the subsidence curve, or 50 per cent. of maximum subsidence. With regard to point **2**, subsidence will be equivalent to the extraction of two-thirds of 5 zone-sections or 3·33. Point 5 subsidence is reduced, therefore, by 5—3·33, or 1·67 effective zone-sections, i.e. 6·3 per cent., giving the maximum subsidence of point 5 as 50—6·7, or 43·3 per cent. The relative values for point **1** are 5—2·5, or 2·5 effective zone-sections, or a subsidence rate of 50—13·2, or 36·8 per cent.

The subsidence rates for sub-areas *c* and *d* have been found in the same manner and the data summarised in the table at the top of the following page.

Since the subsidence rates listed in this table refer only to the extraction of one-half of the full area, and the other half (left side of the diagram) is usually mined simultaneously, the values in the table have to be doubled. The following table gives the total subsidence from full extraction as a percentage of the maximum for each shaft point in each quarter:

Point	*Quarter*			
	1	2	3	4
1	43·0	73·6	92·0	100·0
2	55·0	87·4	100·0	100·0
3	73·6	100·0	100·0	100·0
4	100·0	100·0	100·0	100·0

Point	*Data*	*Period of Full Extraction*			
		1st Quarter	*2nd Quarter*	*3rd Quarter*	*4th Quarter*
1	Effective zone-section . .	1·25	2·5	3·75	5·0
	Points on graph	5 − 3·75	5 − 2·5	5 − 1·25	5 − 0
	Subsidence as percentage of maximum .	50 − 28·5 = 21·5	50 − 13·2 = 36·8	50 − 4 = 46	50 − 0 = 50
2	Effective zone-section . .	1·67	3·33	5·0	5·0
	Points on graph	5 − 3·33	5 − 1·67	—	—
	Subsidence as percentage of maximum .	50 − 22·5 = 27·5	50 − 6·3 = 43·7	50 − 0 = 50	50 − 0 = 50
3	Effective zone-section . .	2·5	5·0	5·0	5·0
	Points on graph	5 − 2·5	—	—	—
	Subsidence as percentage of maximum .	50 − 13·2 = 36·8	50 − 0 = 50	50 − 0 = 50	50 − 0 = 50
4	Effective zone-section . .	5·0	5·0	5·0	5·0
	Points on graph	—	—	—	—
	Subsidence as percentage of maximum .	50 − 0 = 50	50 − 0 = 50	50 − 0 = 50	50 − 0 = 50

In order to complete the analysis, the time-factor must be considered, since the subsidence-time relationship has not been included. Observation records show that in many instances the time-factor has the following effect:

Time	*Percentage of Total Effective Subsidence*
After 1st Quarter . .	50
After 2nd Quarter . .	75
After 3rd Quarter . .	90
After 4th Quarter . .	95
After 5th Quarter . .	100

It is obvious that in the first quarter points 1–4 do not subside to the full extent shown in the previous summary. In addition, maximum subsidence will only be completed four quarters after the extraction of area *d* is terminated, thus extending over a period of two years. The final values for the extraction of the full area when proceeding from the shaft with diverging extraction development are as follows:

Percentage of maximum subsidence for shaft points 1–4 *during eight quarters,* 1–8, *of the effective extraction period.*

Point	1	2	3	4	5	6	7	8
1	21·5	47·6	70·9	87·0	95·3	98·7	99·8	100·0
2	27·5	57·5	79·8	92·1	97·0	99·7	100·0	100·0
3	36·8	68·4	86·0	95·6	99·3	100·0	100·0	100·0
4	50·0	75·0	90·0	97·5	100·0	100·0	100·0	100·0

Fig. 82 shows the values in the above table graphically. The set of curves represents the subsidence of the four points in the shaft and thus shows the effect of the subsidence on the shaft as a whole. The

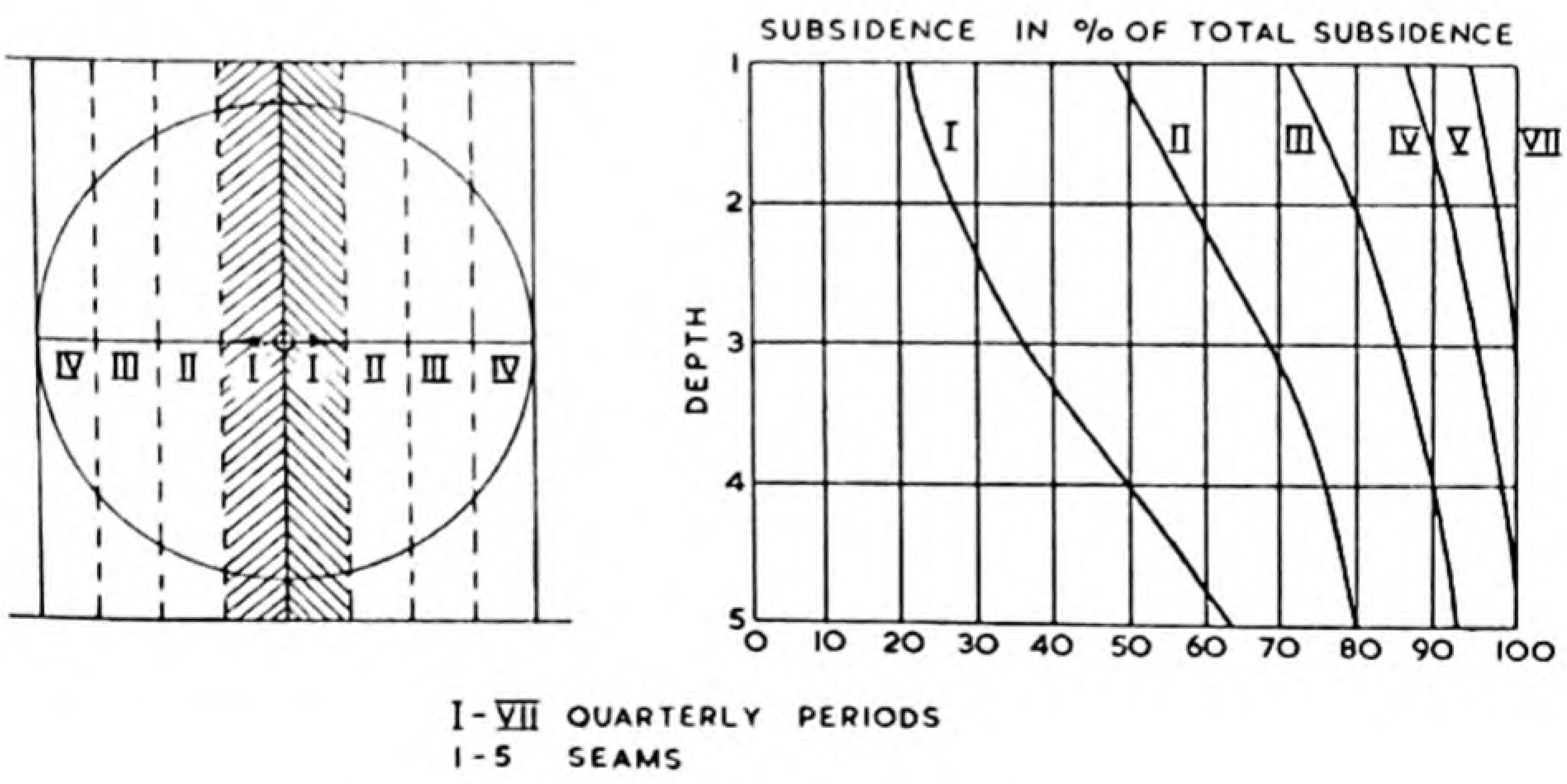


FIG. 82

curves show that the shaft is subjected to a change in the intensity of the vertical tensile forces acting on the shaft during extraction. The vertical tension is a maximum at the commencement, gradually decreasing to zero from the surface downwards, until finally the shaft subsides within the tensional zone of the maximum subsidence. To

illustrate the degree of tension involved during the process of extraction, a seam 6 feet thick, worked with pneumatic stowage, may be considered. The maximum subsidence is of the order of 50 per cent. of the seam thickness, i.e. 3 feet. After the first quarter, therefore, the subsidence in the seam horizon is 3 feet; at point 4, 50 per cent. of 3 feet., i.e. 1 foot 6 inches.; at point 1, 21·5 per cent., i.e. 7·75 inches. The total change in the length of the shaft is therefore 3 feet less 7·75 inches, i.e. 2 feet 4·25 inches, over a total depth of 400 yards. During the next quarter that amount gradually decreases and is finally 0 in the eighth quarter (two years after the commencement of extraction). The fact that only vertical stresses are considered in this method of calculation appears to be justified, as experience has shown that changes in the horizontal stresses are closely connected with the stresses produced in the vertical direction. In zones where the strata are subjected to tensile stress, compressive stresses are also present. The lateral and vertical tensile stresses are related to each other, such that if one is small the other is small and as one decreases so the other decreases. It has been found that these stresses partly neutralise each other, so that vertical tension is diminished by horizontal compression and horizontal tension by vertical compression.

Finally, it should be added that any deviation of the shaft from the vertical can be avoided if the shaft is located at the centre of the full area of extraction.

(*c*) *Various methods of applying the compensation-stress technique to shaft-pillar extraction.* In addition to the method of extraction described and illustrated in Fig. 82, three further systems can be applied to extract a shaft pillar by utilising the compensation technique.

(i) Extraction from one edge of the shaft pillar towards the shaft, and from the centre towards the opposing edge, *vide* Fig. 83.

(ii) Commencing the extraction in two zones, either to the left or right of the shaft, with diverging extraction development towards opposite edges of the pillar, *vide* Fig. 84.

(iii) Utilising a chess-board layout of extraction panels, so that all the faces advance in the same direction, *vide* Fig. 85.

Figs. 83, 84 and 85 show at various periods of extraction the distribution of the faces in plan and the subsidence curves for the four shaft points 1–4, as has been previously discussed.

Consider these systems in sequence. If extraction advances from

one edge of the shaft pillar towards the centre and, simultaneously, from the centre towards the opposite edge, vertical tensile stress is small in the upper section of the shaft, and only slightly greater in the middle and lower sections of the shaft. Vertical compressive stress will be set up due to the face approaching from the side. The curve to the left in the third quarter in Fig. 83 shows a vertical compressive stress. The slope of the curves in the later periods indicates a decrease in this compressive stress as extraction procerds. In the second system of extraction, the faces are commenced in two adjacent zones on one side of the shaft as shown in

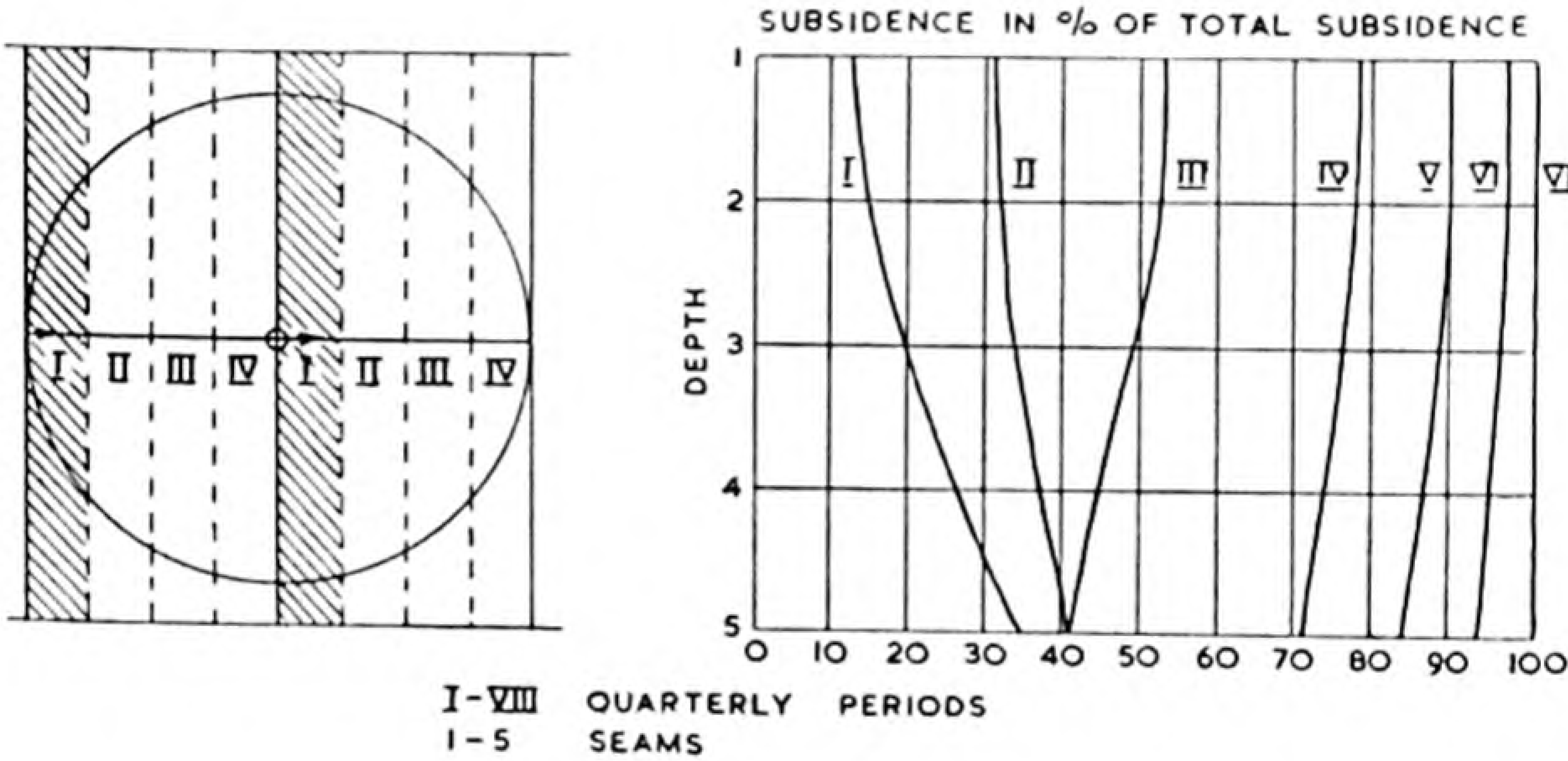


Fig. 83

Fig. 84. The subsidence curves to the left show that the vertical compressive stress occurrs at the middle and lower sections of the shaft. This compressive stress is replaced by a vertical tensile stress in the later quarters of extraction, gradually extending to the upper section of the shaft and showing no tendency to decrease after the extraction has passed through the shaft.

The most favourable conditions for pillar extraction and the control of subsidence is effected by staggering the faces in a chessboard pattern, *vide* Fig. 85. The straight subsidence curves show that vertical tensile and compressive stresses are not produced and therefore no horizontal movements occur. Since the strata subsides evenly at all depths, the shaft subsides also at an even rate. Apart from the intersection of the seam and the shaft, no precautionary measures are necessary.

A comparison of the four methods described shows that the chessboard development is the most favourable face layout. It should be stressed, however, that both the calculations involved and the practical application of the system are difficult, as intensive development work is necessary. The layout in Fig. 86 also requires more initial development work than the system illustrated in Fig. 84 in which

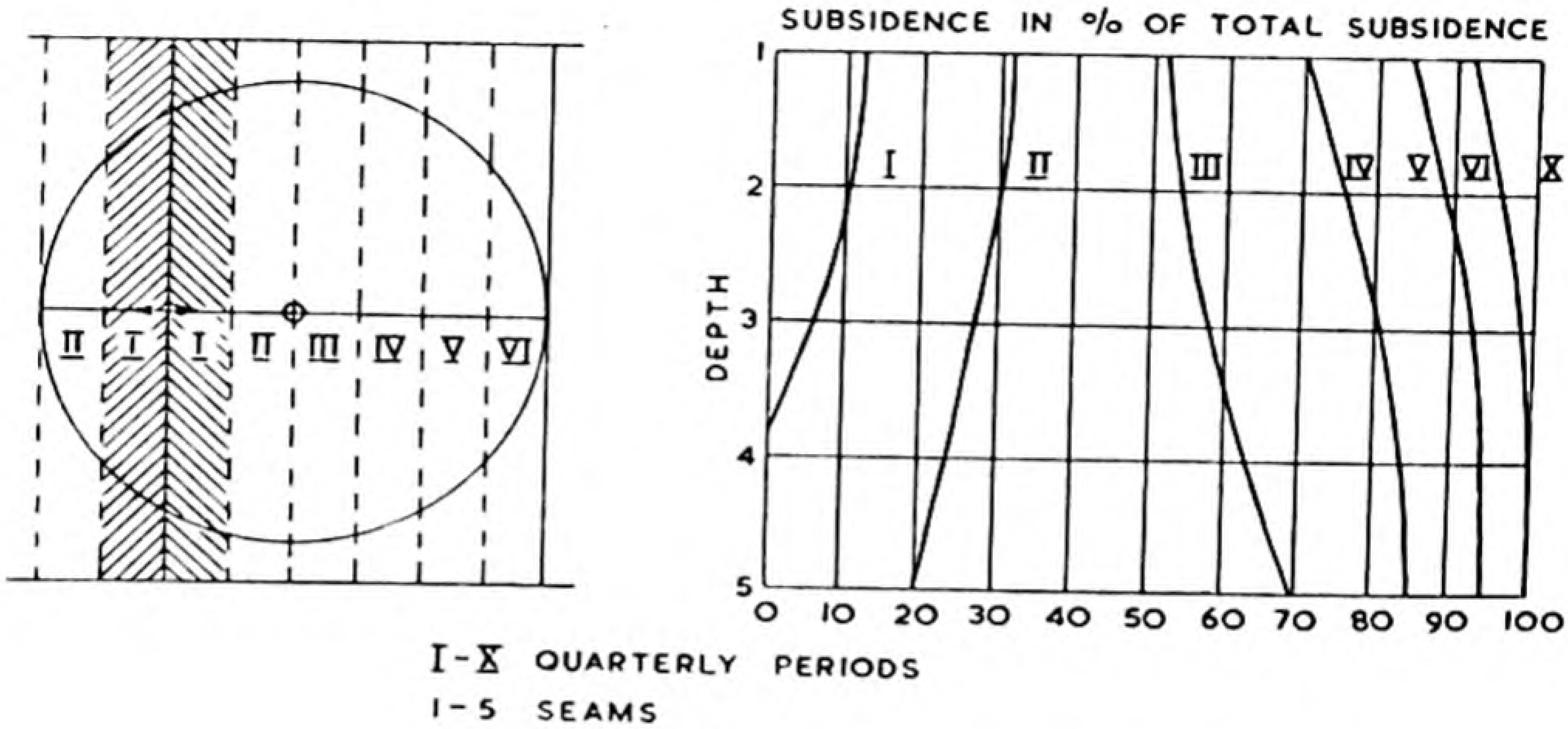


FIG. 84

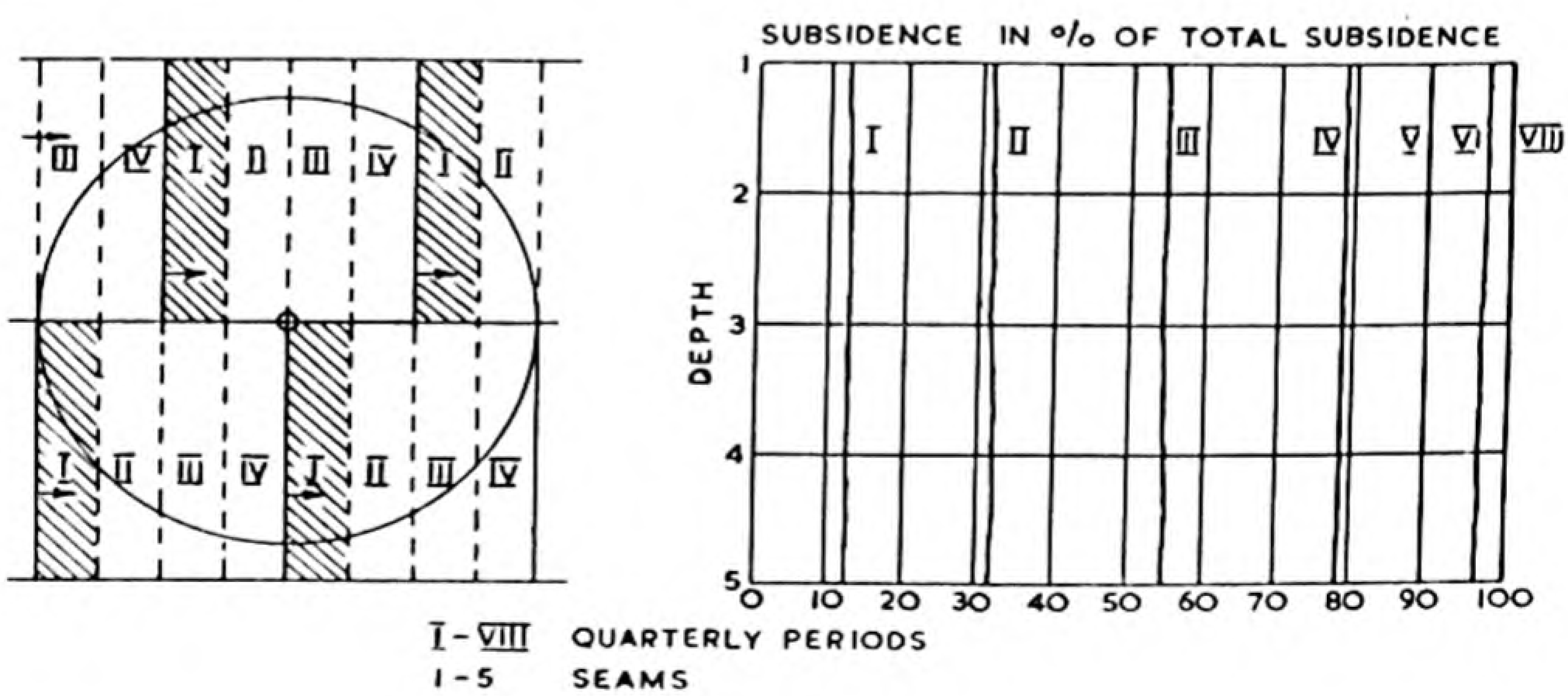


FIG. 85

the face advance is only from one side of the shaft. This latter method has the disadvantage of producing an alternating condition of vertical compressive and tensile stress.

The method normally preferred, therefore, is to work diverging

faces from the centre on each side of the shaft. In shafts up to a depth of 650 yards, only the upper portion of the shaft will be slightly affected, while with deeper shafts the upper section will be affected to a considerable, though not critical, extent. Difficulties encountered during such periods are due mainly to the infiltration of surface water in the downcast shaft. The icy conditions, which may arise in cold weather, make it advisable to introduce a shaft heating arrangement before extraction of the shaft pillar commences. Greater difficulties are experienced in deep shafts, as the full area of extraction is so large that mining cannot be carried out in one period and only a

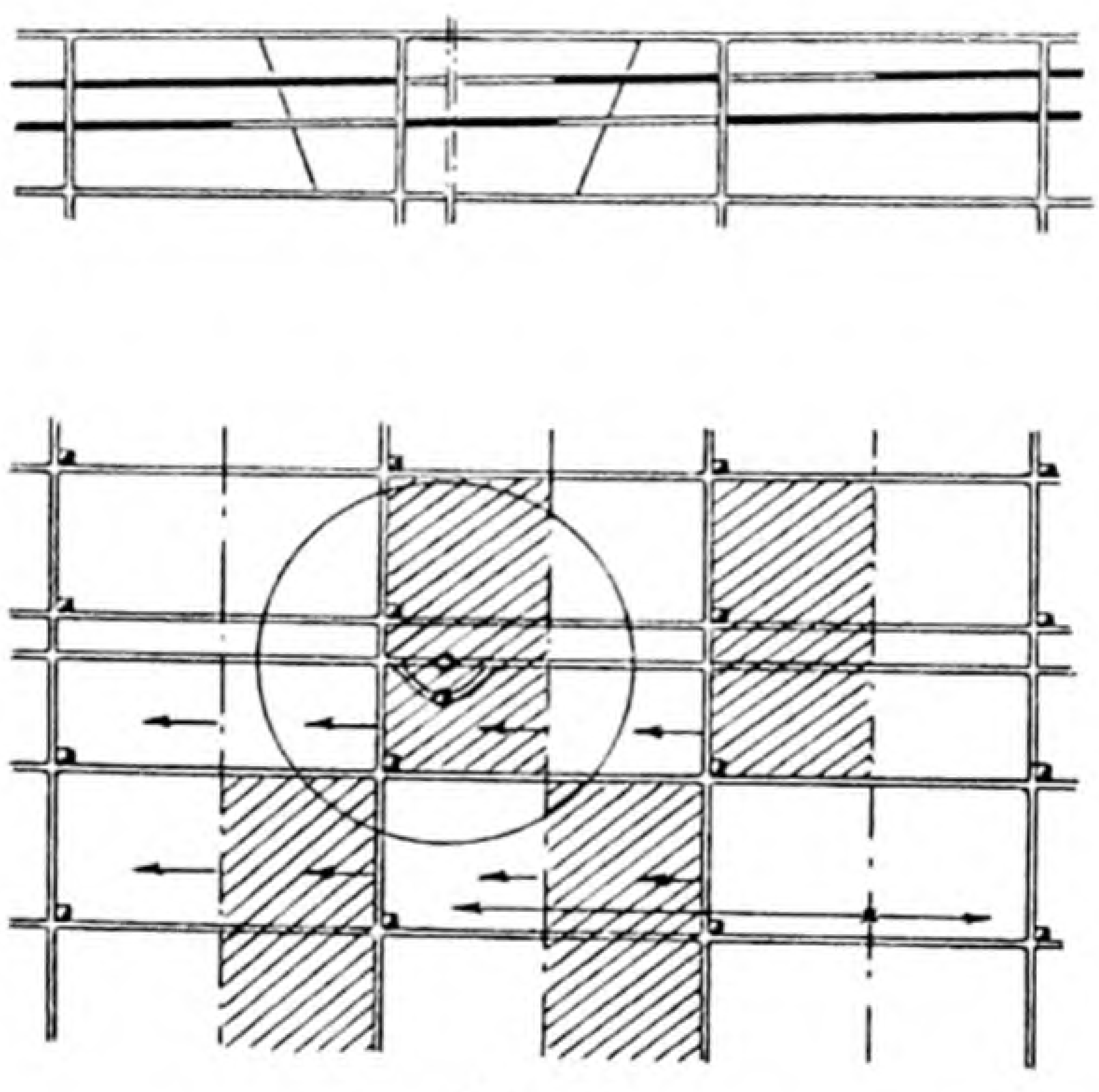


Fig. 86

partial area can be worked. Since the shaft is central and extraction has to be commenced from the shaft, on each side, towards the pillar edges, the area of extraction has to be as large as possible.

In conclusion, it is considered that extraction is possible in the case of all brick-lined shafts which do not penetrate a great thickness of water-bearing measures or quicksand and, in such cases, the shaft pillar should be left intact.

In bricked shafts, however, precautionary arrangements are required, such as special support for seam insets. Brickwork may be replaced by closely packed timbers and wooden packs, behind which the coal has been extracted to a distance of from three to five yards, and the space carefully stowed. Stone dust may also be packed next to

the coal edge to avoid spontaneous combustion. In other sections of the shaft, special wooden inserts may be used in the shaft lining to absorb crush and prevent damage to the shaft wall as shown in Figs. 87A and 87B, which illustrate the conditions before and after working a seam which intersects the shaft at the section shown. The pipelines in the shaft can be fitted with special stuffing boxes, able to take up any vertical movements, while adjustable guide connection

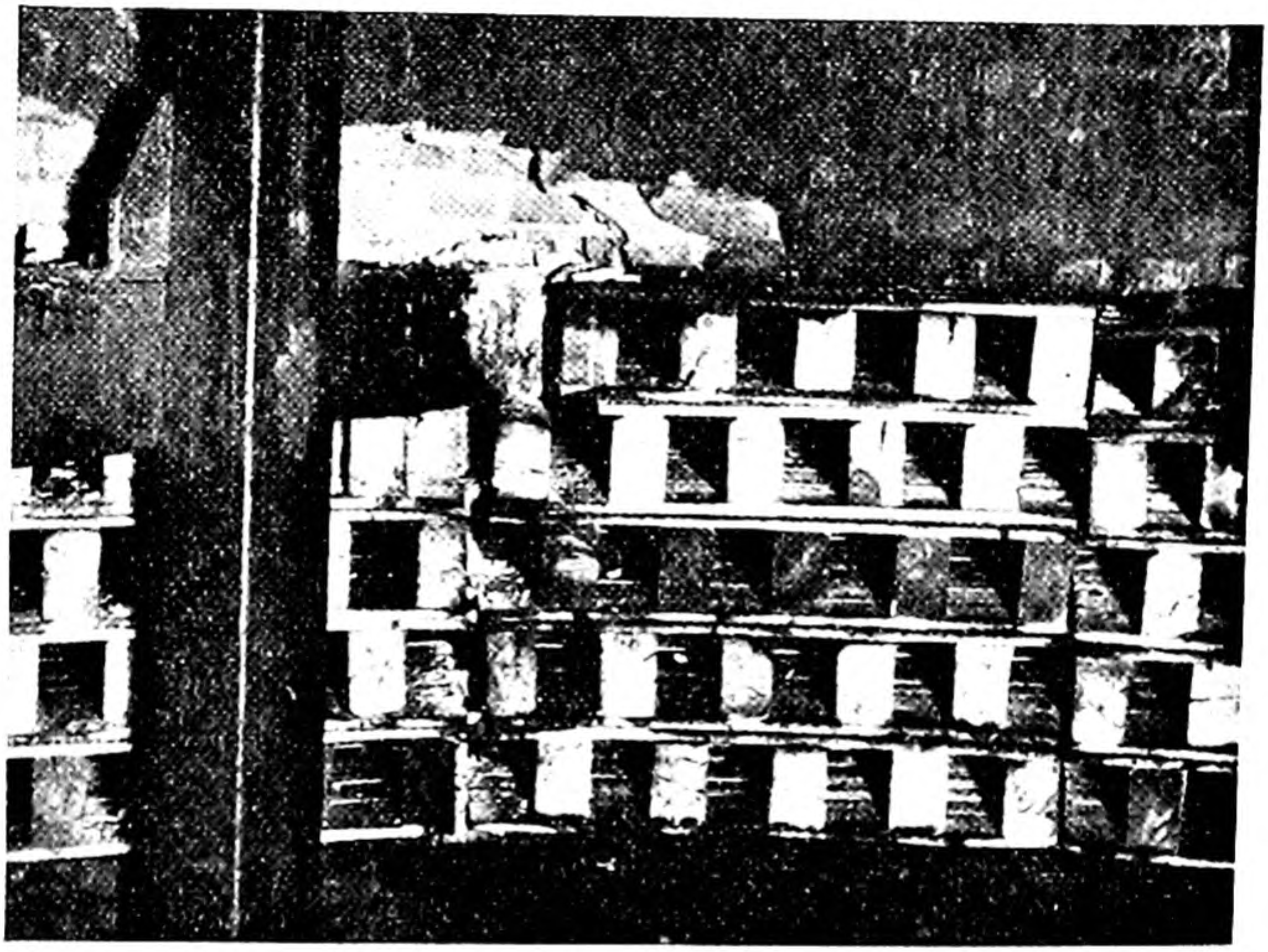


FIG. 87A

plates have been found to be an advantage. Brickwork damage can normally be repaired without great difficulty.

It may be necessary under certain conditions, where a seam or seams have been worked and the shaft pillar left has proved to be inadequate, with the result that the shaft has been damaged by the influence of local stress, to attempt compensation technique to nullify or diminish the effect of this stress on the shaft. This may be done by sequential working of another seam or seams within the zone of the shaft pillar, utilising the technique described.

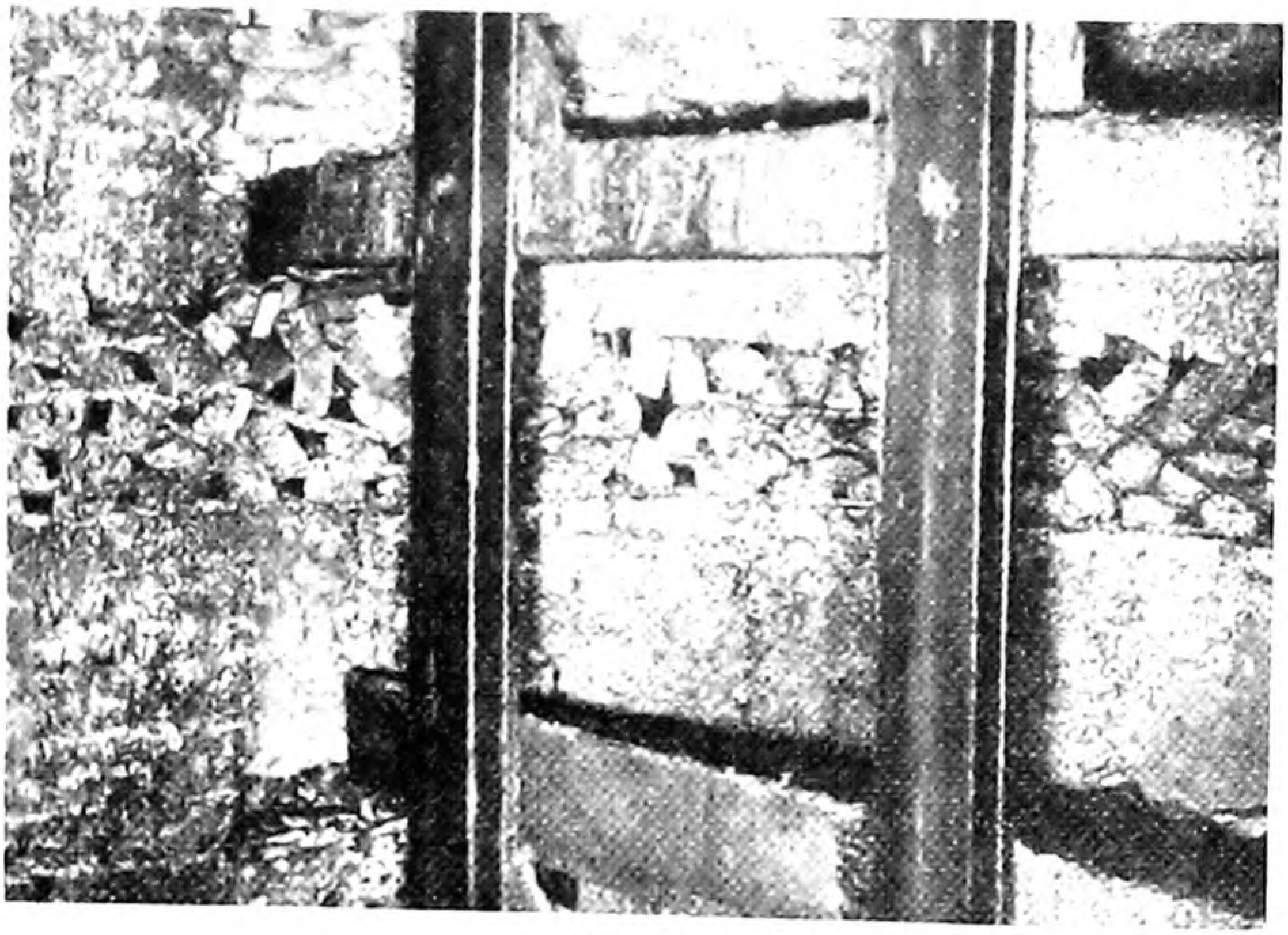


FIG. 87B

Section 4. The Influence of Coal Pillars

(*a*) *Introduction.* Coal pillars can be defined as limited areas of coal seams which have been left during the normal extraction of the seam and around which the coal has been extracted on at least three sides. This condition may have arisen due to various reasons, such as the stopping of a face, due to operational troubles, before the boundary is reached. The seam may have deteriorated in quality, or a band may have thickened to such an extent as to make the seam uneconomic to work. Such pillars can also be part of a general system of development, such as, in the bord-and-pillar, or room-and-pillar, method of working, or where a pillar has been left as protection for a main road.

In all these cases, changes occur in the stress distribution in the roof and floor strata as well as within the coal pillar itself. The effect of this redistribution of stress can produce unequal subsidence.

(*b*) *The effect of the stress changes above and below the coal pillar.* The examination of concrete foundations has proved that the load carried on these foundations is propagated downwards into the ground through the foundation in a solid cone formation, *vide* Fig. 88.

The perception can be applied to mining conditions, as has been confirmed by numerous measurements and observations made underground. Four zones have to be distinguished below a remnant pillar, as shown in Fig. 88. Different intensities and directions of stress are produced within these zones. In zone 1, the induced stress is the greatest and is acting in all directions; within zone 2, the stress is directed outwards and is less in intensity than in zone 1; while within the surrounding lateral zones 3 and 4 in Fig. 88, the stress remains constant. The apex angle of the cone enclosing zone 1 is known as the 'edge angle'. This angle is more acute in resistant types of strata and the cone is deeper. In weak measures the edge angle is greater,

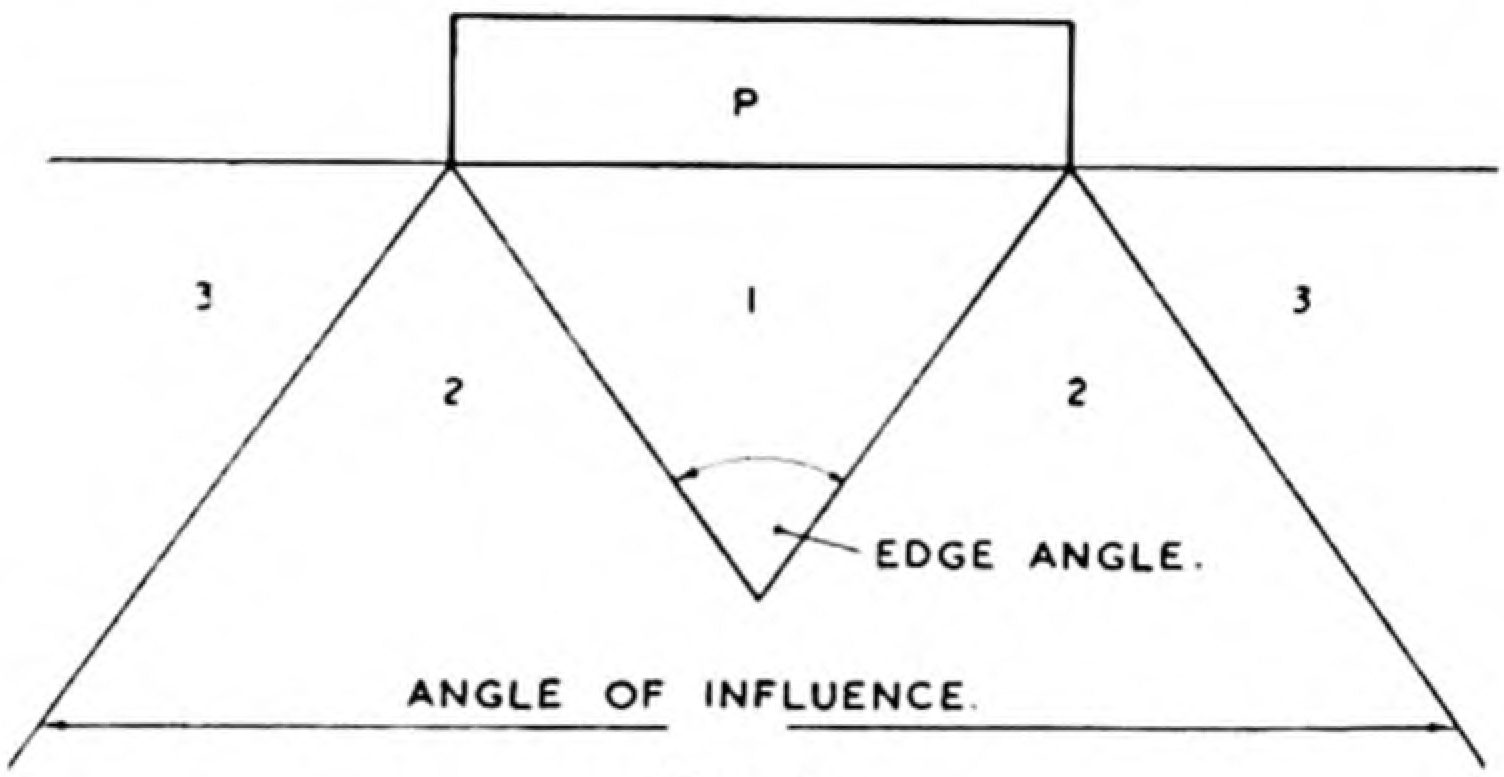


FIG. 88

and the length of the cone is less. The 'angle of influence' must be distinguished from the edge angle, the former being contained between the outer edge of the cone and the line between zones 2 and 3. The magnitude of this angle depends upon the inclination of the seam as well as on the thickness and sequence of the strata layers comprising the floor. Where the strata is level and the stress is transmitted vertically downwards, the angle of influence is identical on each side of the pillar, as shown in the sectional diagram Fig. 88. The angle varies between 60 and 75 degrees. With steep inclinations the angles on the rise and dip sides of the pillar are different. On the rise side the angle is greater than on the dip side, so that the affected zone is restricted on the dip side and increased to the rise, *vide* Fig. 89. Above the coal pillar a similar area of stress exists, which can be divided into a central zone and two lateral border zones, as illustrated in Fig. 90. After the roof of the adjoining goaf has subsided, the edge angle is greater than in the floor distribution, so that the angle of

influence becomes smaller and the affected zone slightly wider in extent. It is therefore possible to deduce that the size of the affected zone, resulting from the presence of the pillar, depends upon the depth below the pillar or the height above it. Similarly, the stress

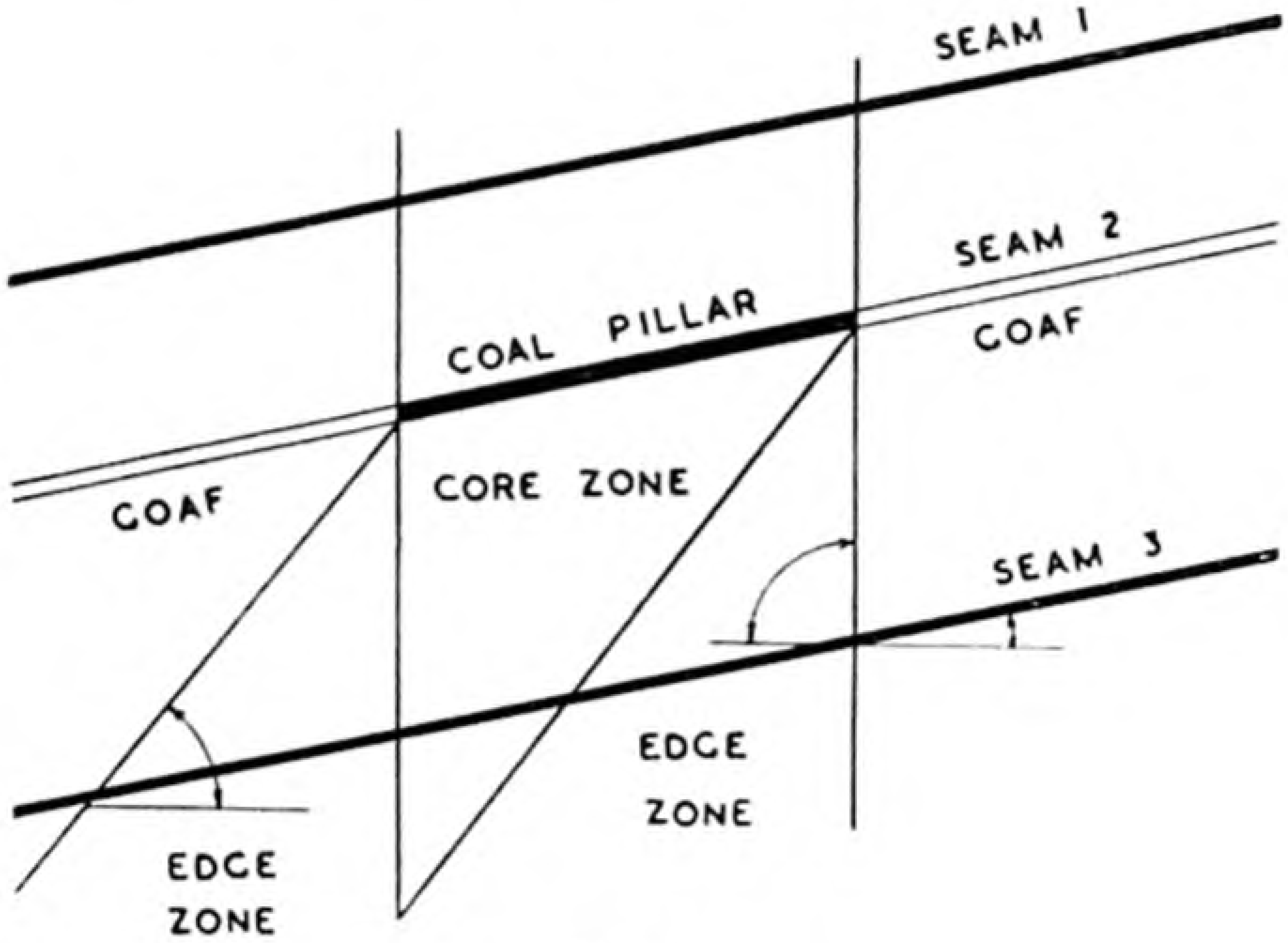


FIG. 89

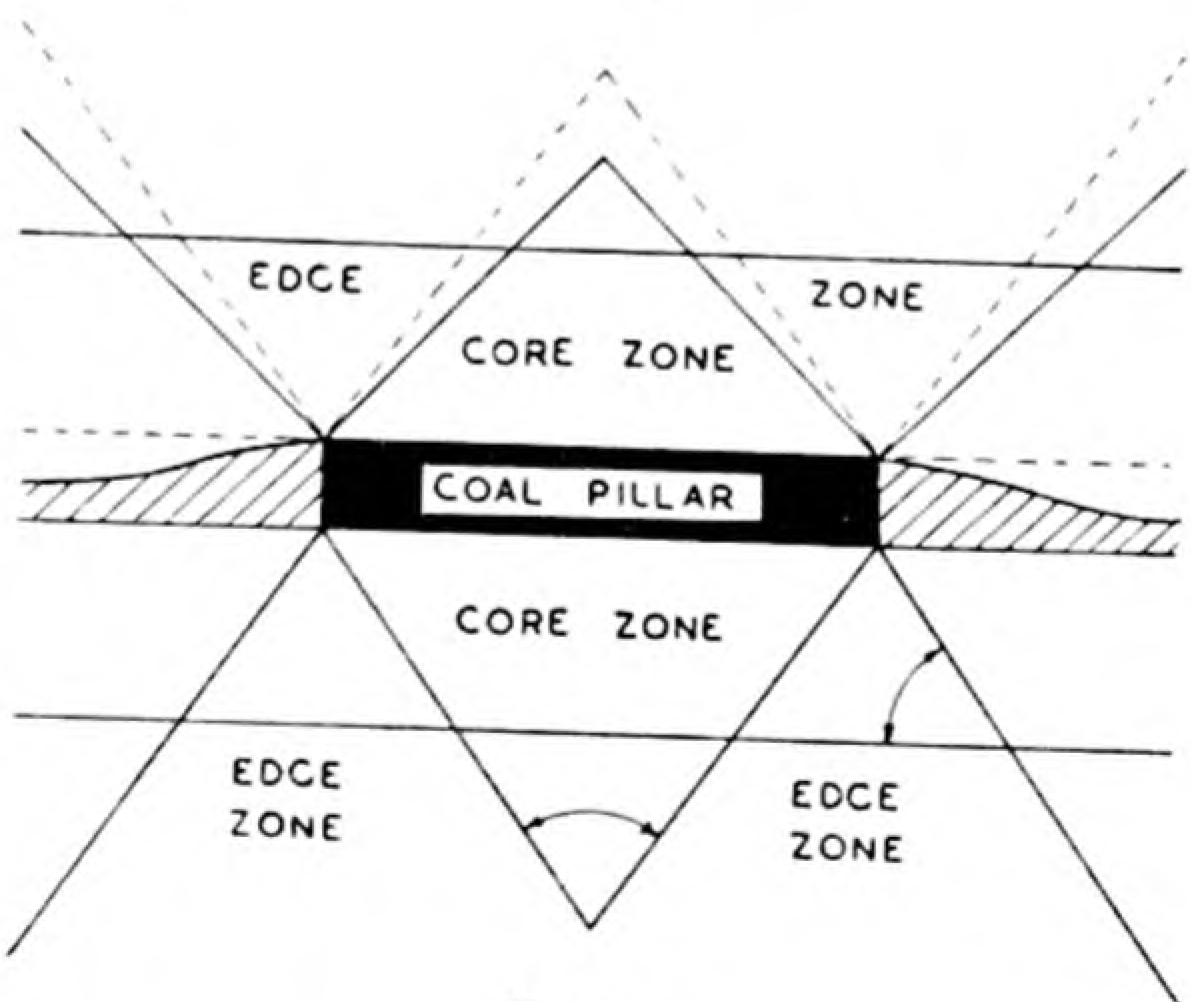


FIG. 90

decreases with distance above or below the pillar. If a seam or stone drift exists slightly below the coal pillar, the core zone in which the stress is high acts over a greater distance than in the edge zone in which the stress is less and the zone is limited in extent, *vide* Fig. 90. With increasing depth below the coal pillar, the area affected by the core zone decreases and that of the edge zone increases, *vide* Fig. 91.

At still greater depths, the influence of the central zone 1 ceases and that of the border zones is extended. The stress decreases with depth, although it may still remain an important consideration. The effect

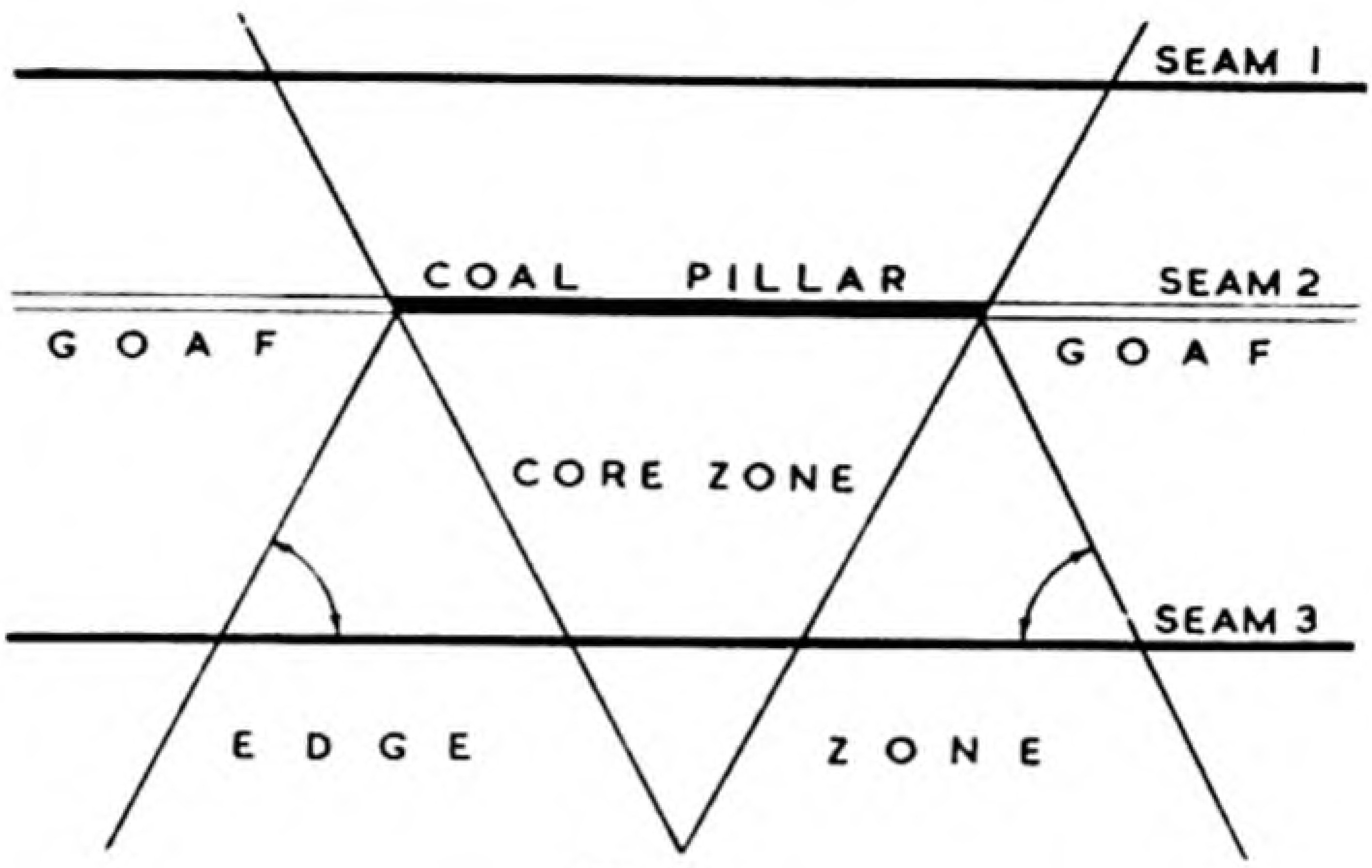


FIG. 91

of the increase in stress has often been observed to disappear after a depth or height from the pillar of from 70 to 90 yards. If, however, there are affected zones from several pillars, situated vertically above

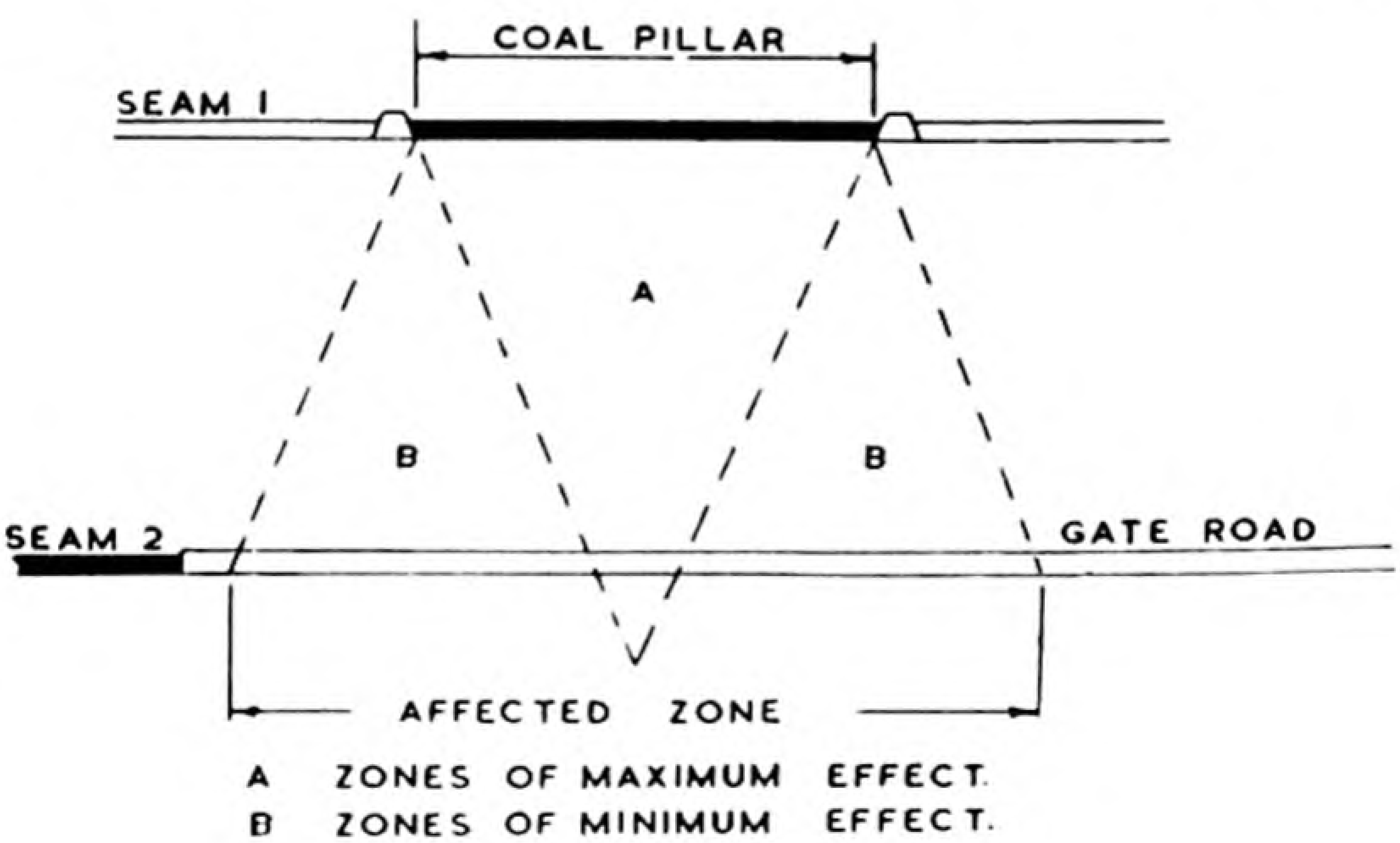


FIG. 92

each other, the effect is cumulative, and the affected zone may reach as far down as 350 yards from the lowest pillar.

An example is illustrated in Fig. 92, which shows a longitudinal section across a 10 yards × 20 yards coal pillar. A mothergate in a

lower seam is 30 yards below the pillar. Outside the affected zone, maintenance on the roadway was normal, while in that section within the central zone shown in the diagram the maintenance costs were high and there were frequent falls from the roof and sides, while the creep on the floor was excessive.

(c) *Stress distribution within the coal pillar.* Since the pillar represents a solid abutment, it follows that it offers the greatest resistance to the weight of the superincumbent strata. Zones of increased pressure are set up along and parallel to each side extending laterally to a distance of perhaps 30 to 40 yards or more towards the centre of the pillar. These zones overlap since they are parallel to the sides of the pillar, and introduce areas in which a cumulative effect is present. Since the pillar could be surrounded by goaf or waste on three or four sides, special stowage measures have to be adopted where the pillar is being extracted. In the case of a pillar worked out on three sides only, retreating extraction from the pillar towards the remaining coal side is preferable, since, with advancing extraction, the last rib of coal remaining in the pillar will be exposed to a high load concentration in advance of the face, making it difficult to work.

Where the pillar is rectangular in shape, extraction should be commenced from the narrower side. Proceeding from the narrow side, the existing increased pressure zones are being penetrated at right angles and the front abutment area set up by the pillar extraction extends over a narrow front, so that the extraction of the remaining coal rib is only slightly influenced by the front abutment loading in advance of extraction.

The development and maintenance of roadways driven in coal pillars also presents a problem. The problem is greater with decreasing size of pillar, since the increased pressure, both above and below the pillar, is dependent on the pillar size. Generally, the effects of the increased pressure round the pillar will be felt when the pillar has reached a length of 300 yards and a width of 200 yards. The nature of the roof also has an important bearing on the stress distribution within the pillar, a sandstone roof being better than a shale roof for reduction of pressure effects. Faults will not interrupt the formation of the zones of higher loading caused by the pillar, but will influence the stress distribution and any stress compensation due to variation in the strata layers.

(d) *Conclusions.* Because of the additional stress set up within the

coal pillar itself and in the strata above and below the pillar due to extraction around it, coal pillars are to be avoided. Pillars are also not recommended for main roadway protection, except where the pillar can be substantial in width on each side of the roadway. Even in this instance, future pillar extraction is only possible at an increased cost.

Protection pillars are often left for all time and are sterilised reserves affecting both the life of the mine and the ability to control adequately the uniform lowering of the surface by a compensating-stress technique.

Thus, complete extraction should be the aim, especially in cases where the working of individual sections means an increase in the working cost. This requirement must be considered carefully in the case of areas where the seam density is high and other seams are present in close proximity to the seam which is being worked.

CHAPTER 3

DEVELOPMENT IN STONE

PART I

DRIVING LEVEL CROSS-MEASURE DRIFTS, LATERAL AND INCLINED DRIFTS

Section 1. *Drilling and Blasting*

(*a*) *Introduction.* The driving of stone drifts can be sub-divided into four main operations: drilling, shot-firing and stripping the face, loading, and putting in supports. Additional preparatory work, such as lengthening track, laying pipes, etc., must also be included in the total time of the complete operation, which will also include unproductive stoppages during repair operations and the normal lost time due to meals. Loading can be carried out either by hand or by machines and, according to which method is used, the drivage is termed 'hand loading' or 'mechanical loading'. The ideal system should be designed to complete the cycle of operations as effectively as possible and to organise the sequence of operations so that the maximum drifting performance is obtained.

(*b*) *The design of the round.* The systematic placing of the shot-holes is of great importance as well as the actual drilling operation itself. The normal procedure is to introduce 'sumping' holes ('cut' shots) in the 'round', which are fired first, providing free faces from which the remaining surrounding holes can act. The number and position of these supplementary holes ('easers and trimmers'), together with their charges, are designed around the first sumping round, and the sequence of firing is denoted usually by round 1, 2, etc. The efficiency of the sumping round determines the effectiveness of the remaining shots, while the depth of the sump determines the advance per round. It is useless and detrimental to drill and fire supplementary holes which are deeper than the sumping round.

(*c*) *Forms of sumping rounds.* The commonest designs for sumping rounds are the pyramid and wedge cuts, though the drag cut, fan cut, and burn cut are sometimes used.

The *pyramid cut* is characterised by the fact that a pyramid-shaped

cavity is fired out with the sumping shots from the rock face. Where fairly light strata is being fired, four holes usually will be sufficient for the round. In strong ground usually eight holes will be required, four of which are drilled to half the depth of the round and the remaining four to the full depth of the lift. With very solid rock a ninth hole is necessary in the centre of the pyramid cut, parallel with the direction of drive; this modified round is called the *cone cut*, *vide* Fig. 93. The holes are drilled at the corners of a square 25 to 40 inches apart. The closer together the holes are drilled, the smaller the

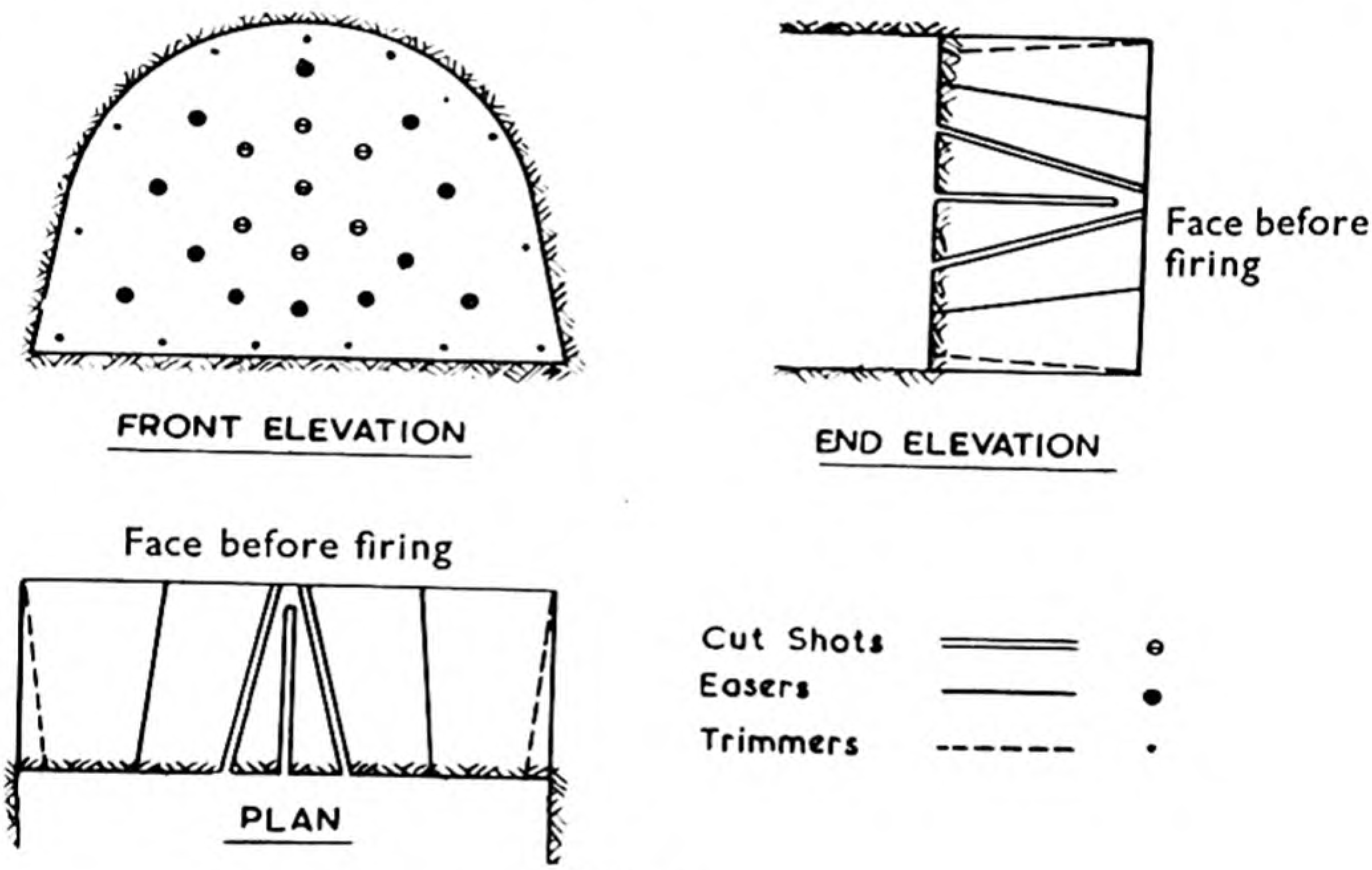


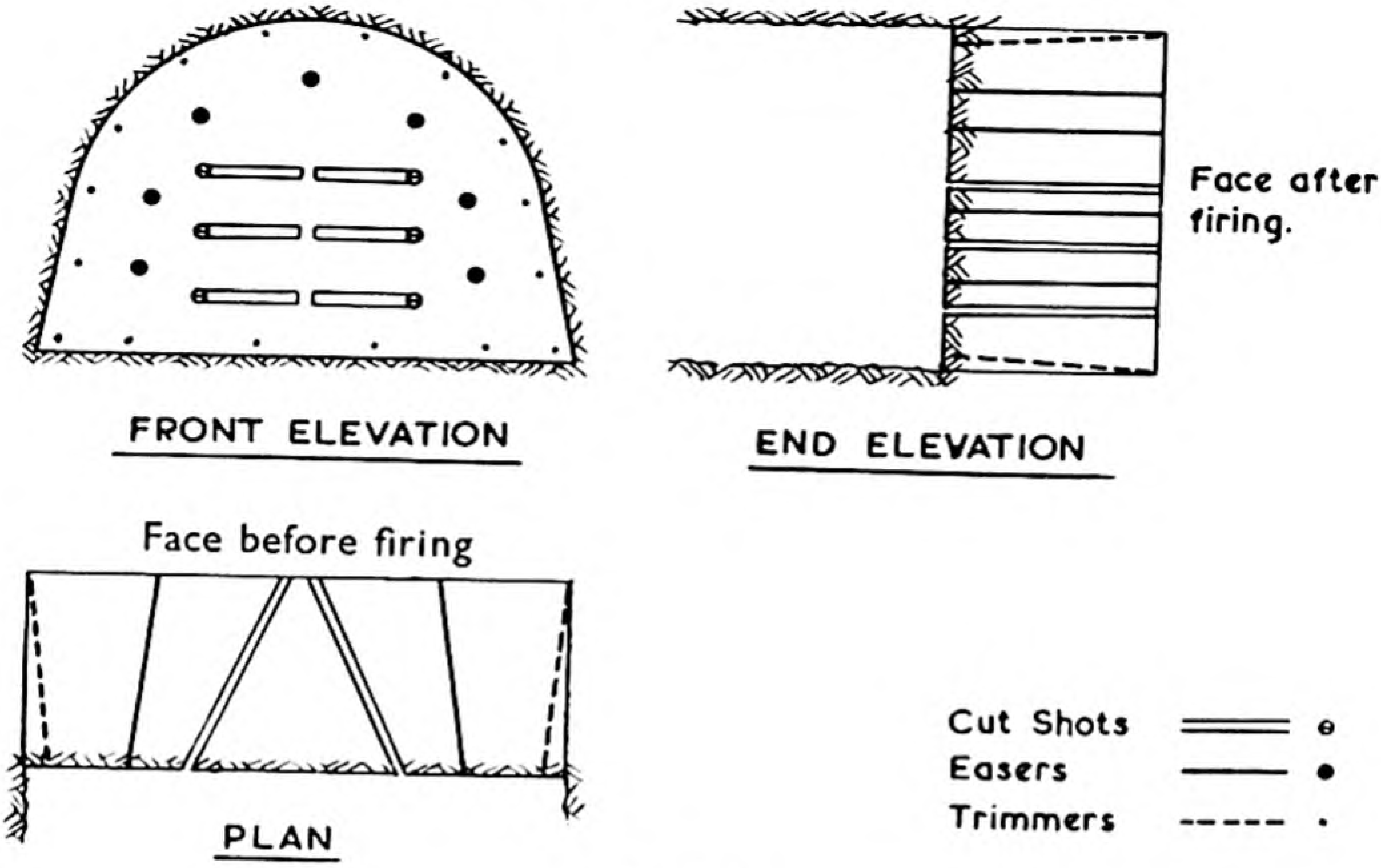
Fig. 93

initial pyramid cavity and the less the possibility of damage to supports and pipe-lines by flying rock fragments. The sumping round should be set exactly in the centre of the drift axis and should not be overcharged. In order to keep the face from spreading away from the sump, the trimmer holes or ring shots at the roof level from the previous round may be left and fired instantaneously with the cut holes of the succeeding round. Spreading of the cut shots can be avoided by charging the two upper cut holes heavier than the lower holes and firing the upper holes with instantaneous delays followed by No. 1 delay on the lower pair. With an eight-hole cut, the inner sump holes are fired before the outer sump holes. The cut holes should

be placed so that they do not touch or intersect at the back of the cut. In the case of the inner, or short, cut holes, they should finish on the corners of an 8-inch square, while the outer sump holes should end at the corners of a 14-inch square. The pyramid cut is used where uniform or thickly bedded rock, which may be either horizontal or inclined, is being drilled. It can be used also with other rock formations, as it can be drilled easily and safely and uses a smaller total explosive charge. The disadvantage of the system lies in the fact that it allows only a definite depth of cut to be lifted per round, depending on the width of the drift. The pyramid cut must be drilled to a depth which will allow a reasonable quantity of explosive to be used, so that the side support is maintained and the amount of rock to be loaded out from the cut is not too great. Generally speaking, the depth of the pyramid cut is kept to within 50 per cent. of the width of the drift although, by using successive outer cut rounds, it is possible to increase the depth of the cut to 60 per cent.

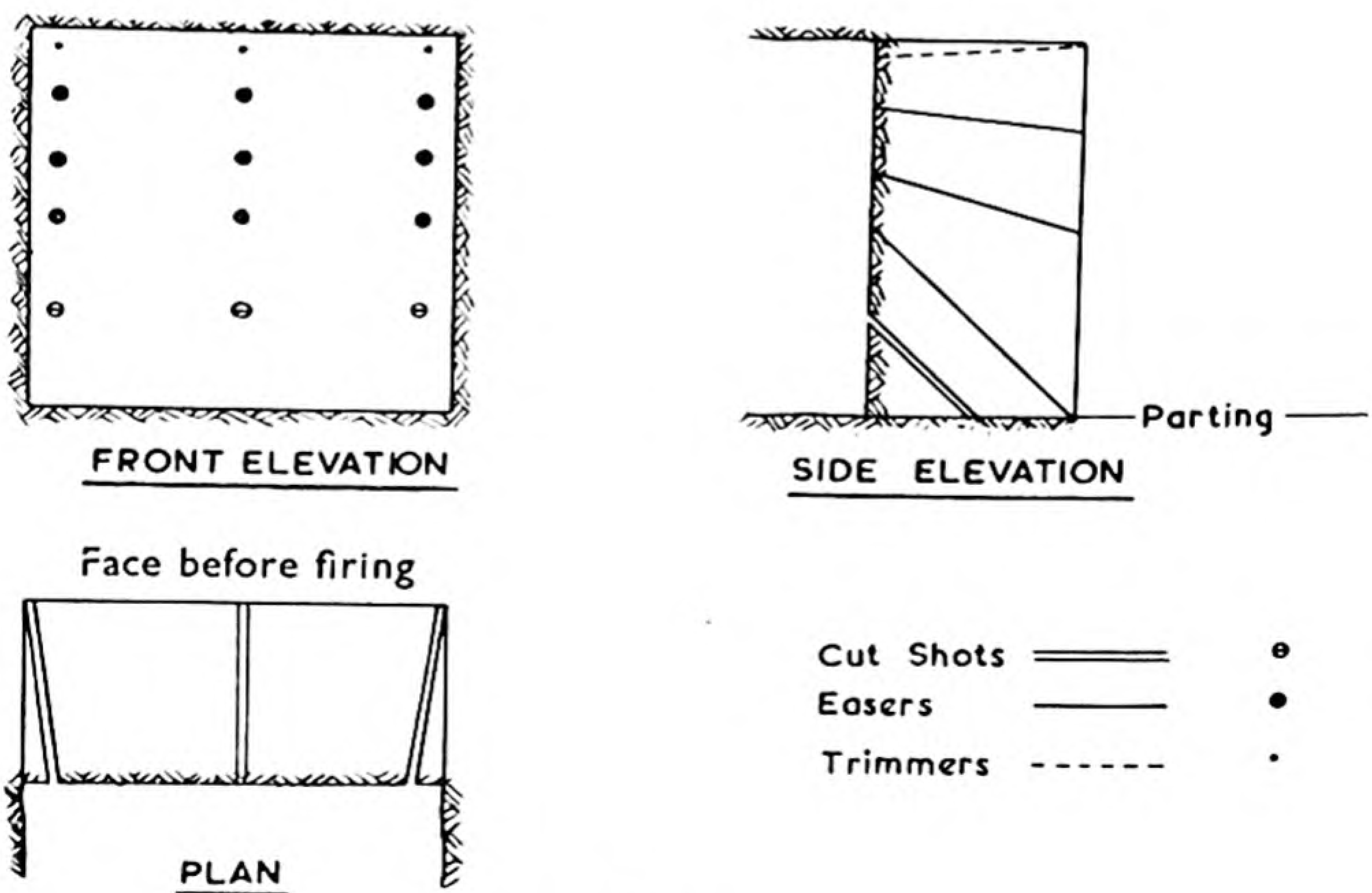
The wedge cut is often called universal sumping. The drilling pattern is in the form of a triangular wedge which is generally vertical but can also be drilled horizontally. Two to four pairs of shot holes are required. Each pair of holes is suitably placed on both sides of the drift centre and flanked inwards at an angle of less than 45 degrees so that two holes will finish close together at the back of the cut, *vide* Fig. 94. This procedure is limited by the width of the drift and the longest drilling rod which can be used. If the rock is a very solid strata, several wedge cuts are used, so that the first pair of holes must be placed at a fairly acute wedge angle, with the second and third pair correspondingly flatter. The advantages and disadvantages of this form of cut are similar to the pyramid cut.

The drag cut is especially suitable for well-cleated rocks which allow the holes to be placed at an angle to the cleat so that the cut breaks along the strata cleavage planes, *vide* Fig. 96. Where the strata is horizontal and the drift is being driven either with the cleat or across the cleat, the use of a top sumping cut or bottom sumping cut is important. With cut holes placed at an angle to the cleat, the strata breaks along the cleavage planes and the subsequent shots can act upon this free face, both upwards and downwards. The cut holes should not be drilled past the cleavage parting, and the number of shot holes and the amount of explosive charge required is small. A disadvantage of this form of sumping is that it depends upon the



FRONT ELEVATION
END ELEVATION
Face after firing.
Face before firing
PLAN
Cut Shots
Easers
Trimmers

Fig. 94



FRONT ELEVATION
SIDE ELEVATION
Parting
Face before firing
PLAN
Cut Shots
Easers
Trimmers

Fig. 95

form and condition of the cleavage planes in the strata and their distance apart, it cannot be used continuously where the rock nature changes and it is possible, therefore, to have varying lengths of lift. For high-speed drifting this system would retard the cyclic progress of the work and is not recommended for large drifts.

The fan cut is not used to any great extent in coal-mining, although favoured in metalliferous mining, since it is only suitable for foliated

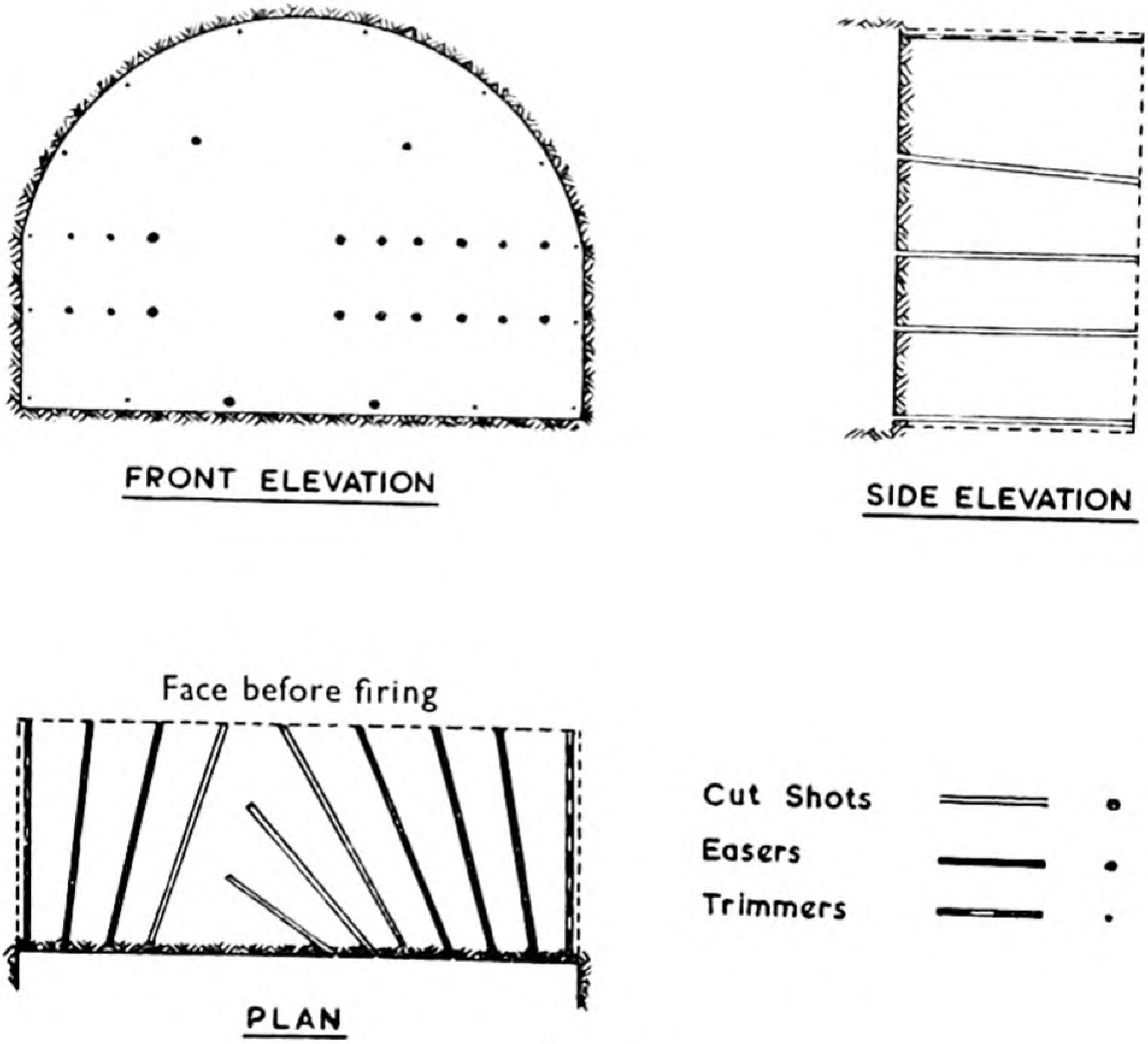


Fig. 96

and well-stratified strata. The holes in this case are drilled to cover the face width with a fan-like pattern in one plane, *vide* Fig. 96, and has the advantage that the spreading of the shots is directed towards the bottom. This cut is suitable for soft and laminated strata, but has the disadvantage that each shot must act for itself, so that the rock to be broken down by each shot must not be too great. For the same reason the total explosive charge is high.

The burn cut has been introduced into coal-mining practice in hard, brittle, homogenous strata, where the rock breaks cleanly. Wide experience in hydro-electric projects, where increased speed of drivage has resulted from the use of this cut, has led to its use and development for stone drifts in coal mines where permitted explosives are used. Parallel cut holes are drilled in a cluster in the direction of the drive at uniform centres. The distance apart varies with the character of the rock and the preliminary tests, but is usually

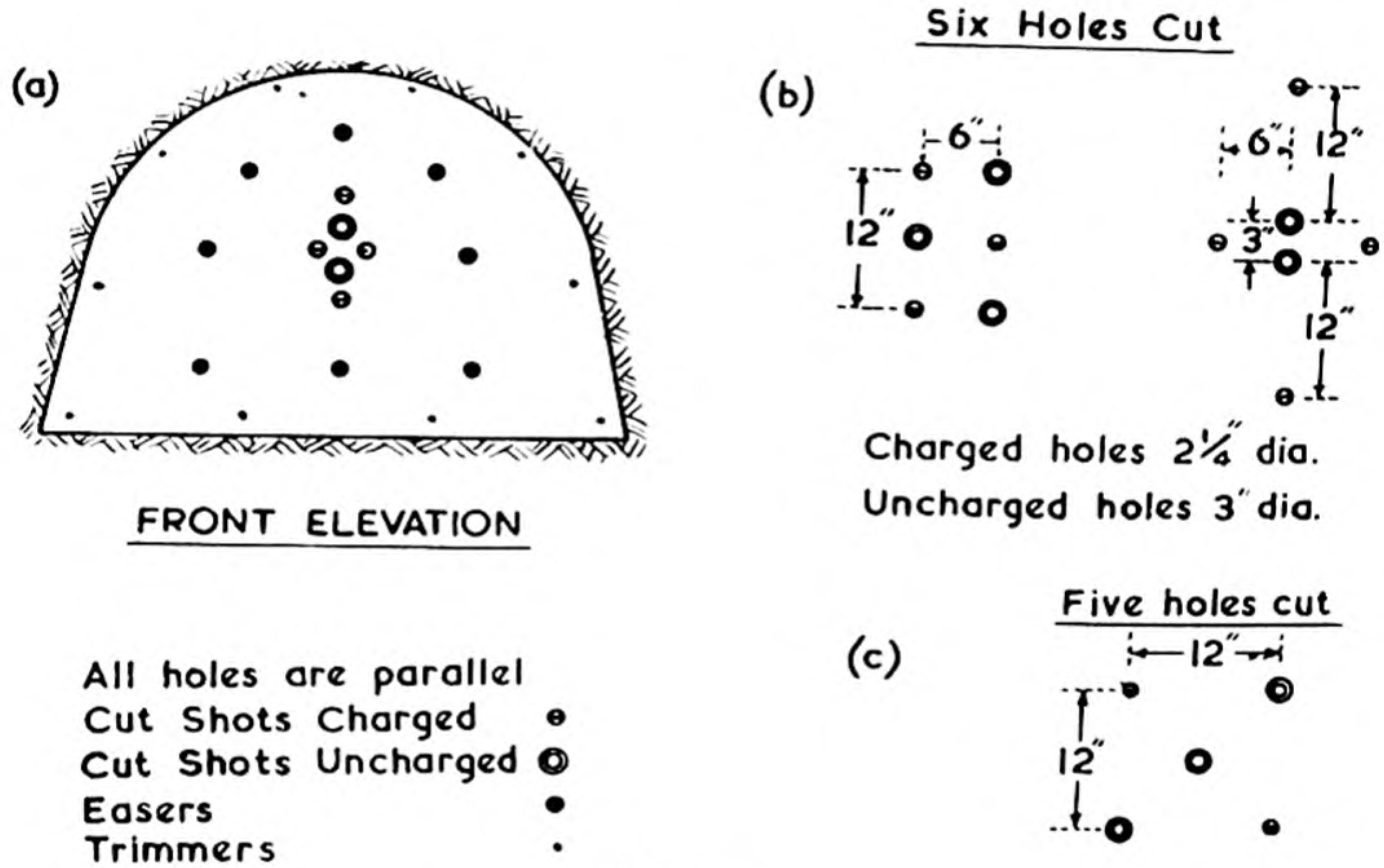


FIG. 97

about 5 inches. Some of the holes are left uncharged to give relief to the charged holes. In some cases, the centre holes in the cluster are drilled at a larger diameter (3 to 6 inches) to give even more effective relief. Applications have been made to the Inspectorate to increase the permitted maximum charge per hole to 48 ounces, with favourable results in the depth of pull in hard strata. The form of the general drilling pattern is shown in Fig. 97(*a*), while various alternative cluster forms are shown in Figs. 97(*b*) and 97(*c*).

(*d*) *Auxiliary and supplementarry ounds.* Supplementary holes are drilled around the cut holes of the sumping round in order to enlarge the cavity broken by the cutters. The holes drilled adjacent to

the sump or cut holes are termed easers and are designed to allow the remaining trimmers to complete the lift to the dimensions of the drift. These auxiliary or supplementary shots are generally charged weaker than the sump holes.

The number of supplementary holes depends upon the cross-section of the drift and the type of strata being fired, the number

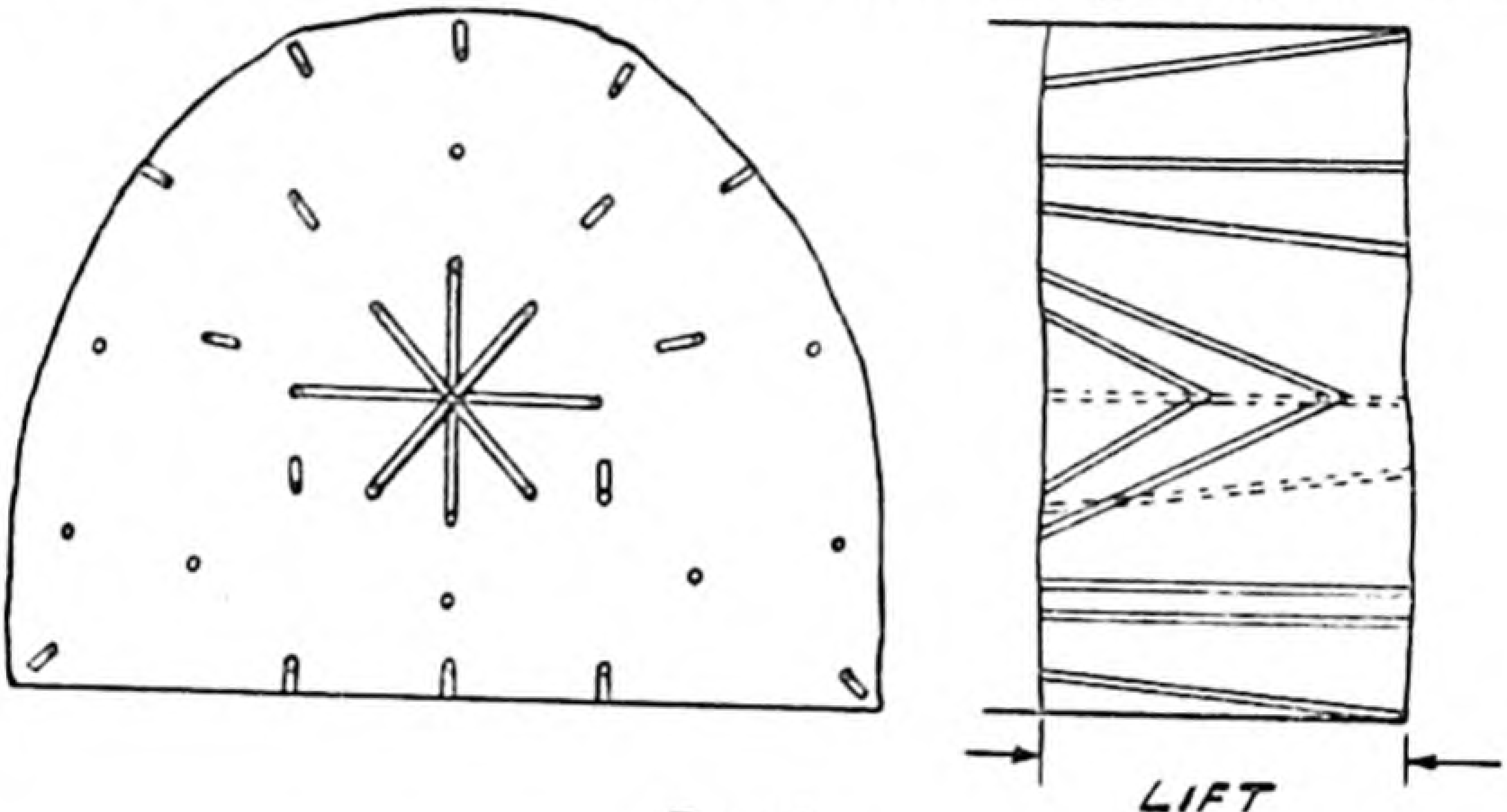


FIG. 98

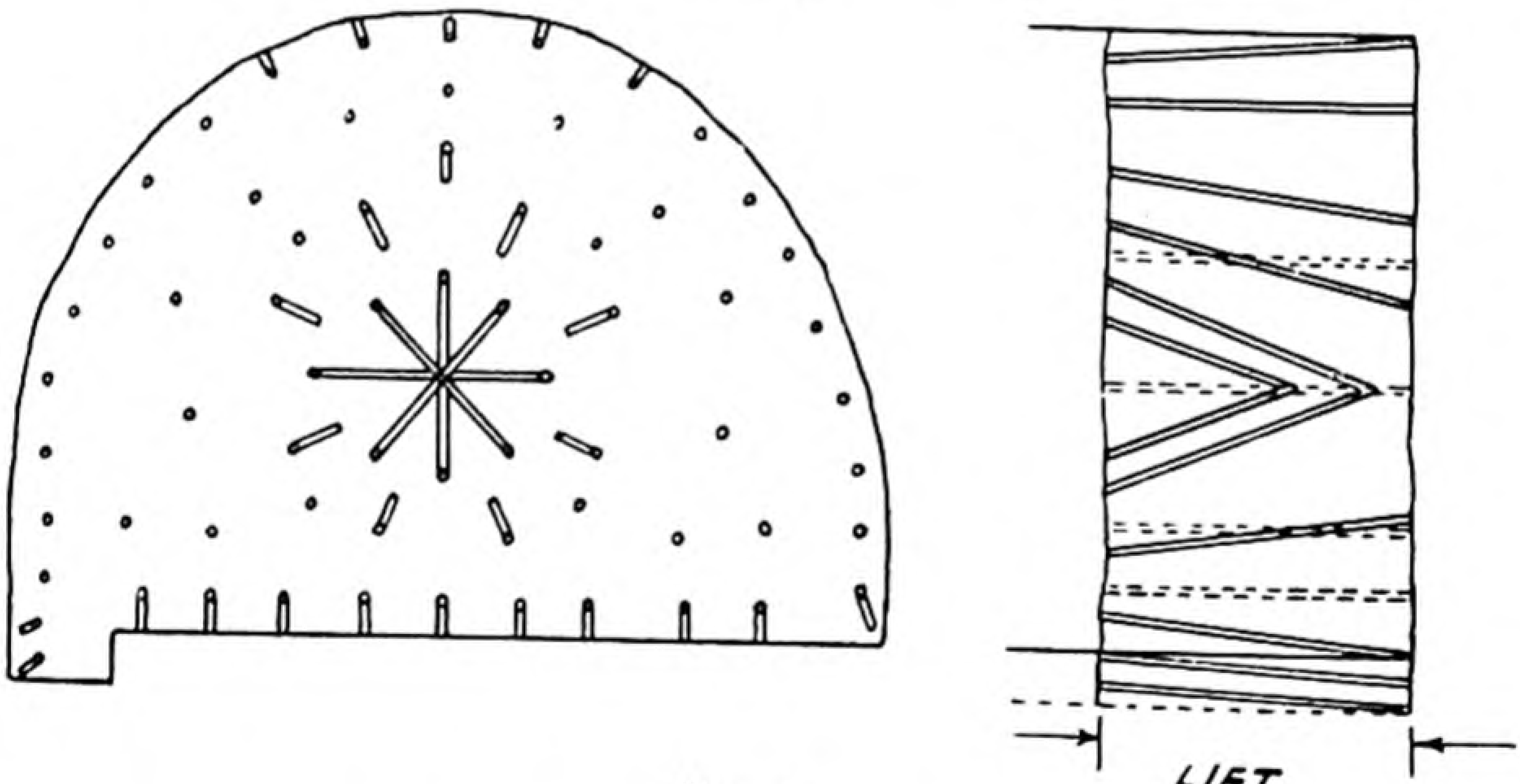


FIG. 99

increasing with the size of the drift and the strength of the rock. It is also important when designing the round to consider the explosive to be used, since the use of a gelatinous-type explosive, as against the normal high-density stone-work explosive, will increase the number of holes required. Fig. 98 shows a drilling pattern for a drift 110 square feet, using eight cut holes and twenty-four easers and trimmers, when rock-drifting explosive was used, while Fig. 99

illustrates the drilling pattern in a drift 150 square feet, using eight cut holes, nine easers and forty-nine trimmers, when a gelatinous explosive was used. The fan cut is illustrated in Fig. 100 in a drift 200 square feet, when nine cutters and forty-eight trimmers were used with a gelatinous explosive charge.

The direction of the top and bottom shots in a round is important, since the drilling of these holes is sometimes rendered difficult by the rock face or supports. These holes should be drilled so that they are as close to the finished drift size as possible, but should not be deflected or 'flanked out' so that the back of the hole is beyond the

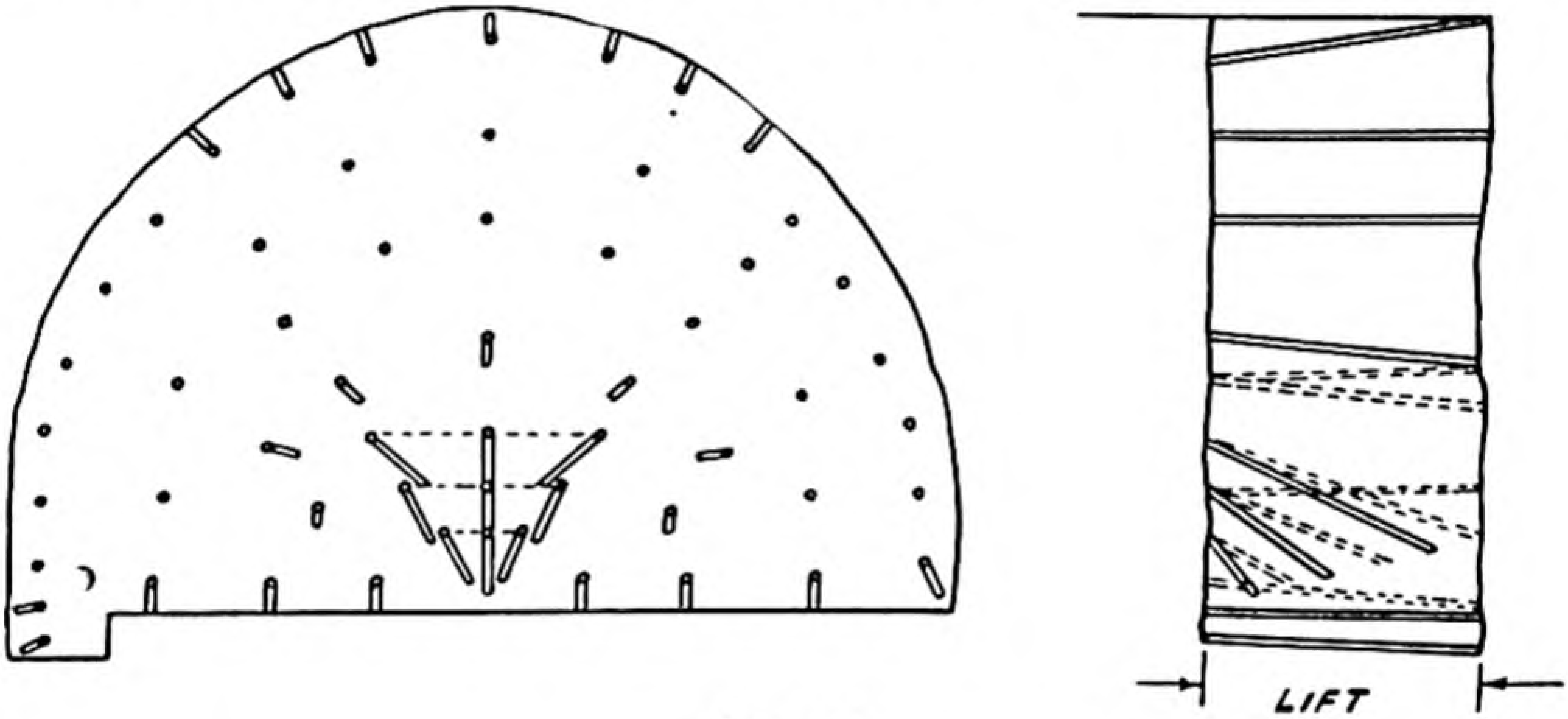


FIG. 100

breaking-line of the drift cross-section. Each enlargement of the section beyond requirements decreases the drifting performance and adds to the amount of waste to be loaded out, increasing both the time and labour required, and making it necessary to lag the spaces behind the permanent supports.

Similarly, the reverse process can have an equally inefficient result when the holes are not flanked out sufficiently to cover the drift size, so that additional shot holes have to be fired to allow the permanent supports to be inserted. In view of these minor but important reasons, it is recommended that the drilling pattern should be fixed by measurement and not estimated by eye. In cases where hand-held drills are being used, the pattern can be measured, using the stemming stick in the direction of the hole. With modern rig-mounted equipment there is more certainty of maintaining a standard drilling pattern, but it is still equally important to see that the holes are drilled to the required pattern both in direction and space.

The location of the shot holes is also of great importance. In normal cases the location of the shot holes depends upon the type of strata to be drilled and fired, and following this principle, holes should never be drilled on bedding planes but into the solid. If the latter precaution is not taken, the resulting waste will break large and be difficult to load out. The shots will act against the bedding planes or slip beyond the desired range, and may influence neighbouring or later shots.

In some instances, where modern high-speed drifting technique is being used and the emphasis is placed on the completion of the cycle

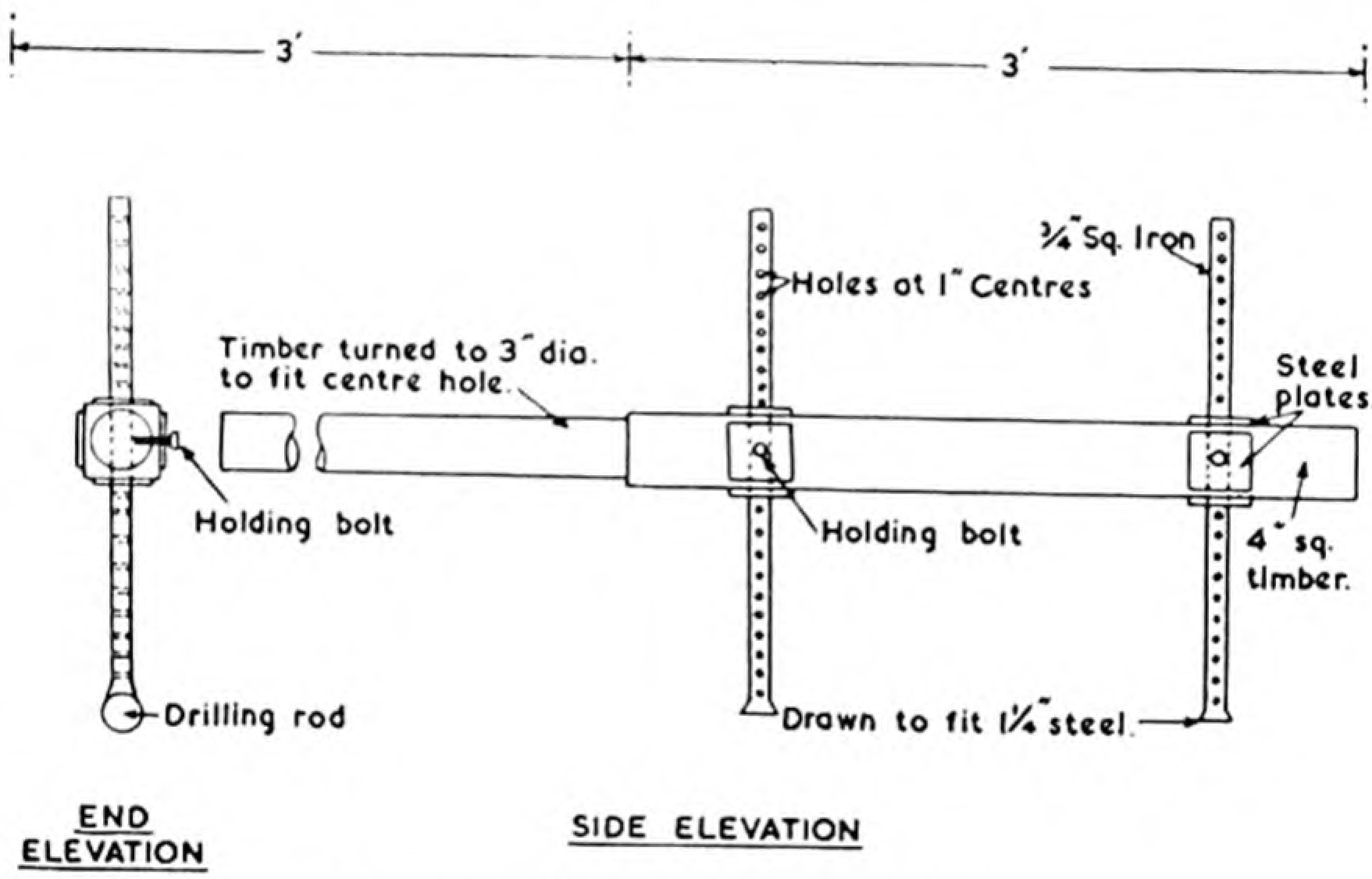


FIG. 101.—Drill guide for the burn cut.

on a definite time schedule, the drilling pattern is set to the best average round for the rock conditions in the drift. The round is maintained at that pattern in order to attain speed both in setting up the drilling rig and drilling and firing the round. A drill guide, template or hole director should be used to ensure consistent results, *vide* Figs. 101, and 102.

Which ever system is used, however, a pattern or design based on experience and maintained by close supervision should be adopted.

(*e*) *The length of lift* or 'depth of pull' is the advance or drivage per round. The longer the lift or pull, the better the utilisation of time, since the accumulation of changes between individual working operations is decreased; the rate of utilisation of the individual

machines used for the different operations will be improved; and that proportion of the productive time taken up for the erection and dismantling of machinery, as well as the waiting time during firing, is substantially reduced.

The advance per round is limited, however, by the depth of the cut, which is itself dependent upon the character of the rock and the sectional area of the drift. The type of explosive used also affects the

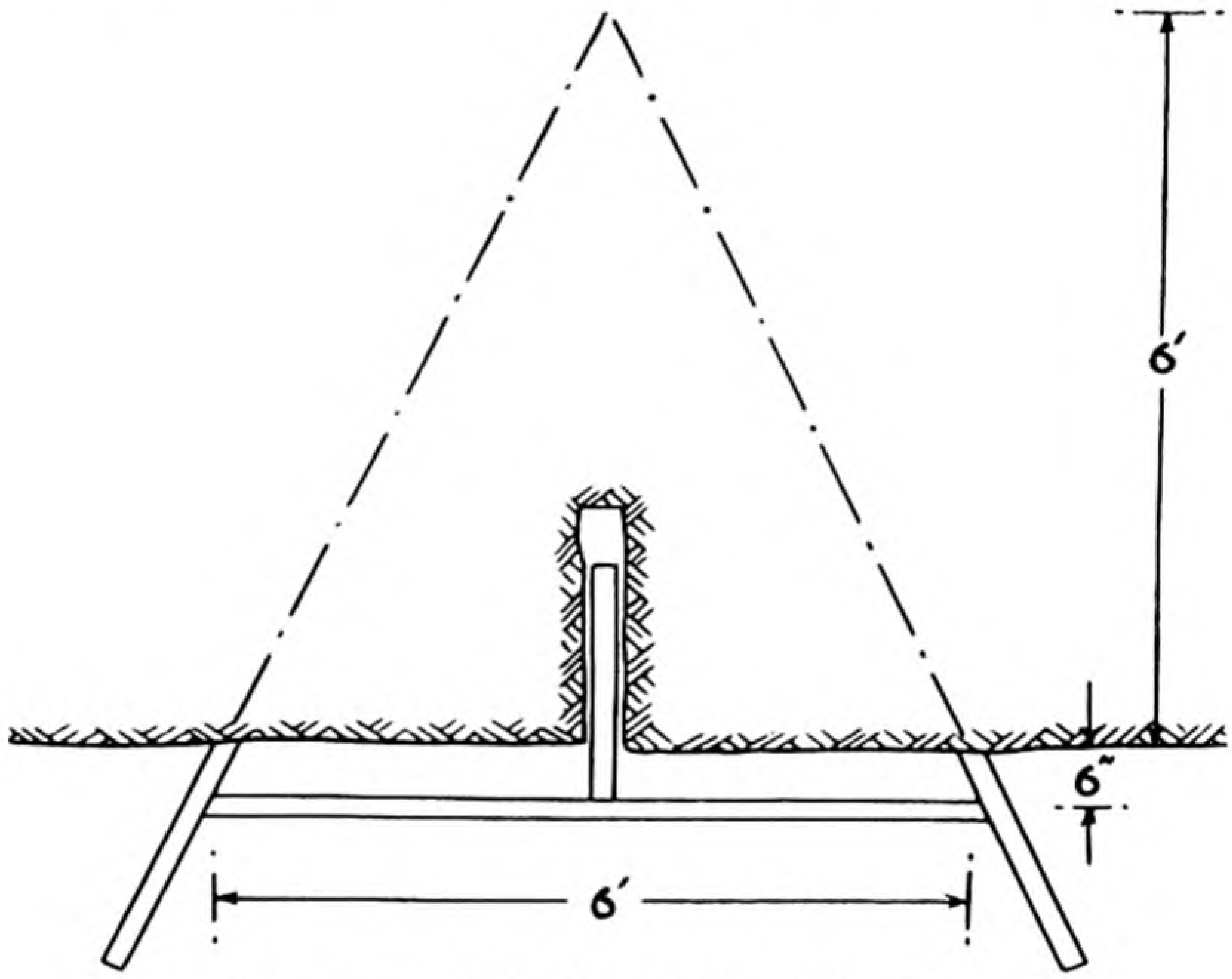


FIG. 102.—Hole director for wedge cut.

pull per round. In drifting, the explosive used should be one of the high-density type, giving high power and high velocity of detonation.

Where there is no danger from gas, an unsheathed explosive should be used, increasing the bulk power and decreasing the diameter of actual explosive used. In normal coal-measure strata, one of the permitted explosives, such as Polar Ajax, unsheathed, can normally be used, although an increase in the maximum charge per hole of 28 ounces may be essential. In the latter case, application must be made to H.M. Chief Inspector of Mines, who will lay down certain conditions to be observed.

Good organisation of the drifting programme is essential for speed and good performance. It is better to limit the pull per round

to a length which will allow completion of the cycle than to increase the lift to a length which allows remaining operations to fall behind schedule. The length of the pull will vary according to the local conditions, but will be between 4 feet 6 inches and 12 feet, the normal pull being about 6–7 feet.

(*f*) *Consumption of Explosives.* The consumption of explosives in stone drifting has not yet reached the same stage of computation by formulae as has been done for quarries or open chamber blasting, as it depends on the influence of many variables. With the normal drift cross-sectional areas between 90 and 200 square feet, it varies from 0·4 to 3 lb. per cubic yard of solid rock, where permitted explosives are used. Fig. 103 shows graphically the explosive con-

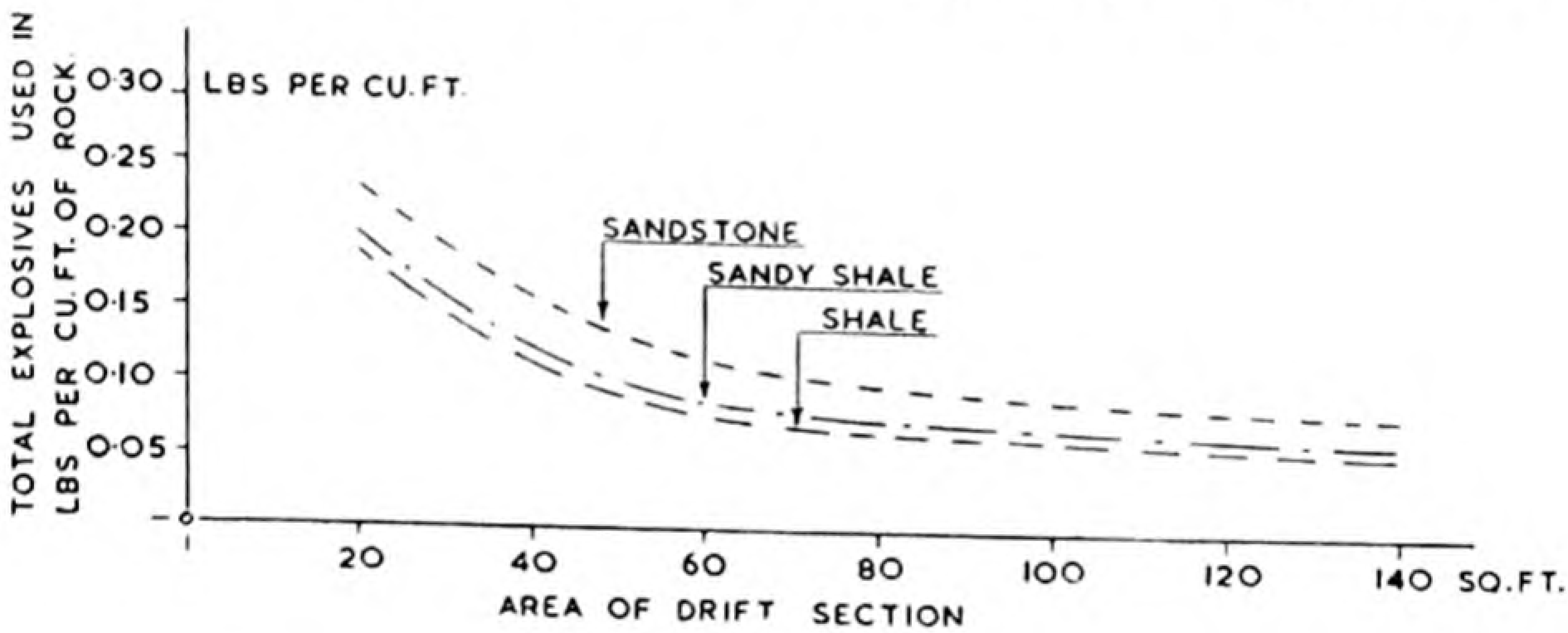


FIG. 103

sumption per cubic foot and per square foot of drift section. It can be seen that the consumption per square foot decreases with increasing cross-section, remaining almost constant after a section of 110 square feet. The curves show that shale and sandy-shale differ only slightly in the consumption of explosives, while sandstone differs widely from these other rocks. The difference decreases with increasing cross-section.

(*g*) *Sequence of firing.* The probable sequence of firing the complete round of holes should be established before drilling and firing is commenced. The firing of a round in stages takes longer, though it probably uses less explosive and a smaller number of holes than where the round is fired simultaneously. In the case of driving by hand-loading methods, it is generally preferred to fire the supplementary shots in one or several groups after the cut has been fired and loaded out. If this is done, it is possible to take full advantage of

the cavity made by the sumping shots, and place the easers and trimmer holes accordingly. The easers will be fired with the sump holes only when a successful sump round is assured; the trimmers are then fired to the best advantage where free faces are present, and the last holes to be fired are the top and bottom corner holes.

When driving stone drifts in the direction of the strike of the strata, special precautions must be taken in the case of stratified measures to ensure that a shot in the trimmer round is not fired before the adjacent shot in the easer round has been fired. The result of such a mistake in the sequence may be a 'blown out' shot or interference within the region of the following shot, producing large, loose material which is difficult to handle, and there may be the possibility of unfired cartridges being left at the back of the second hole which may detonate in the open or be left lying in the waste. Such shots are both wasteful and dangerous.

In mechanised drifting, simultaneous shot firing by the use of delay detonators is a great advantage and has rapidly reached great favour. The explosives cost may be higher, since a greater number of shot holes is used. The decisive factor is the mechanised loading out of a complete lift or pull in one uninterrupted working operation, and the use of a mechanical loading device is justified as it is utilised to its full capacity. The operators are required to stem, fire, strip and load only once per pull.

Fig. 104 represents a round pattern in which the sumping shots are fired simultaneously with the supplementary holes, using delay detonators.

The gasless delay detonators, Fig. 105, at present in use have been developed to enable individual shots to be fired in a predetermined sequence. Delay detonators for use in coal mines are made with a range of numbers from 0 to 10 with a 1-second delay period between each number, and ½-second delays have also been in use. Recently, millisecond-delay detonators with intervals of about 30 milliseconds have been introduced with success. They have the advantage that explosive is saved and the debris is smaller than with other detonators. The use of delay detonators is governed by special regulation for mines working under Part II of the Coal Mines Regulations. At present, when giving permission for the use of these detonators in a gassy mine, H.M. Chief Inspector of Mines stipulates that 'the maximum delay of the highest number shall not exceed 4

seconds'. Fig. 106 illustrates the sequence of firing in a drift in shale strata, using a wedge cut. Joint adaptors or insulators are recommended for use in simultaneous firing with gasless delay detonators to eliminate current leakage and to remove possible danger from stray electric currents. When the drift is wet, their use is particularly

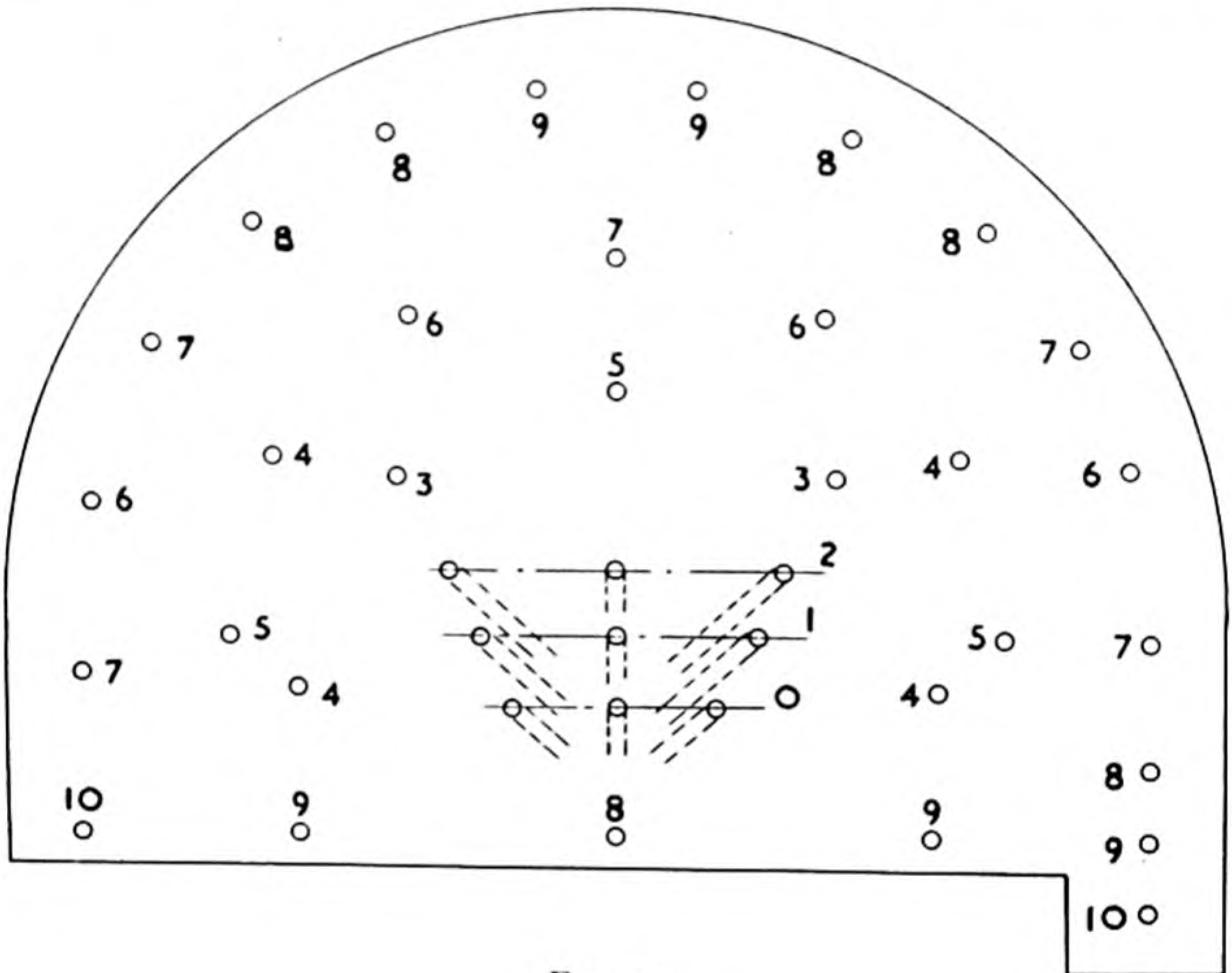


FIG. 104

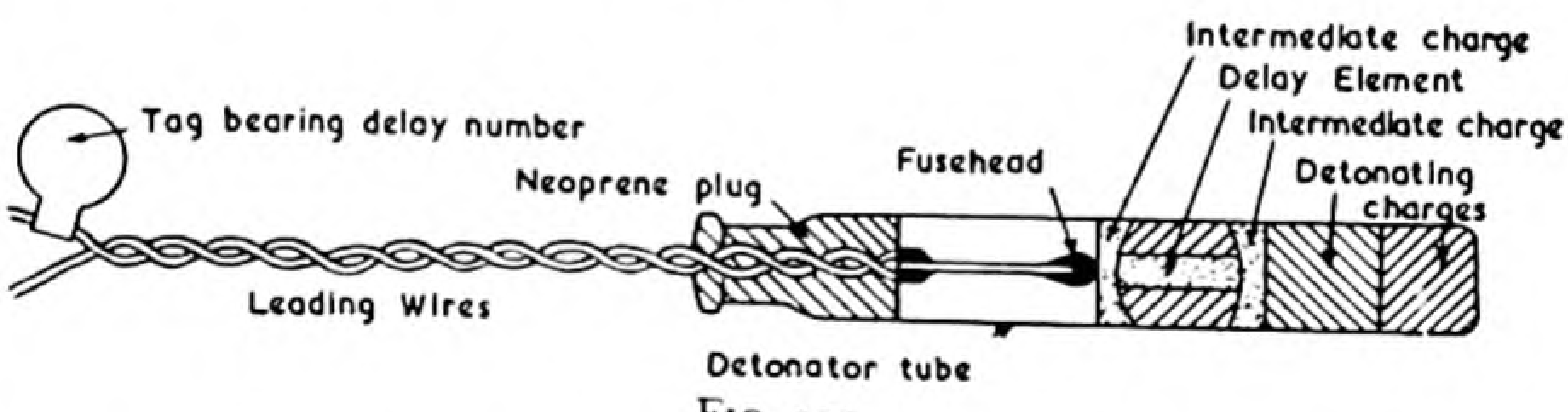
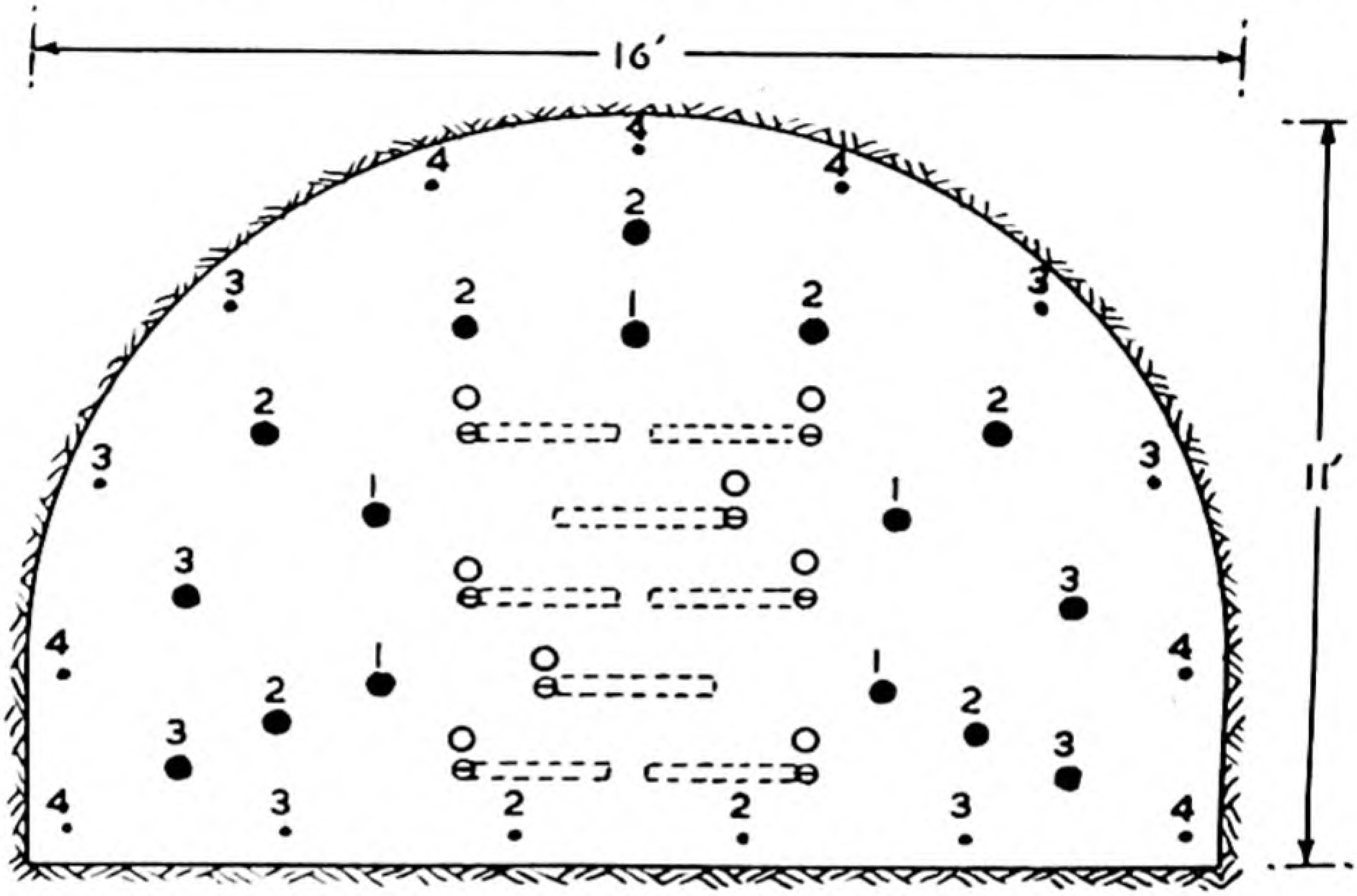


FIG. 105

necessary. The adaptor consists of a cardboard tube, $3\frac{3}{4}$ inches long by $\frac{5}{16}$ inch diameter, filled with bitumen and sealed with cork. The insulator consists of flexible tubing, 3 inches long, $\frac{1}{4}$ inch external diameter and $\frac{1}{16}$ inch internal diameter, with one end sealed with a sulphur plug. When using the adaptor, the cardboard tube is cut in half and the twisted connection of the detonator pushed up one of

these halves until the bare wires are completely buried in bitumen. When making up primer cartridges at the face, the joint insulators are slipped over the bare ends of the leading wires of the detonator to overlap the insulating cover on these wires.

(*h*) *Percussive drilling machines.* The shot holes may be drilled by



		DELAY No.	CHARGE In ozs.
8	Cut Shots	O Instantaneous	28
5	Easers	1	28
7	Easers	2	24
4	Easers	3	20
2	Trimmers	2	24
6	Trimmers	3	20
7	Trimmers	4	20

Total :- 39 Holes . 57 ½ lbs. Pull oer round 6ft

FIG. 106

compressed air, or electrically driven percussive or rotary machines.

The hammer drill is widely used for drilling in mines. It may be mounted as in the drifter type, a hand-held drill or a stoper.

Drifters. The drifter types are heavy machines varying in weight from 100 to 200 lb., while power-feed drifters may weigh even more. The hammer drill is operated by compressed air and consists of a cylinder, piston and striking head, valve-control gear, ratchet and

twist bar to impart rotation to the rod, chuck and water- and air-inlet control connections. The general construction of two drills of this type is shown in Fig. 107, while details of a rotation gear are shown in Fig. 108. The drill is mounted in a cradle of either the fixed or sliding cone pattern. With the latter type the cone is adjustable

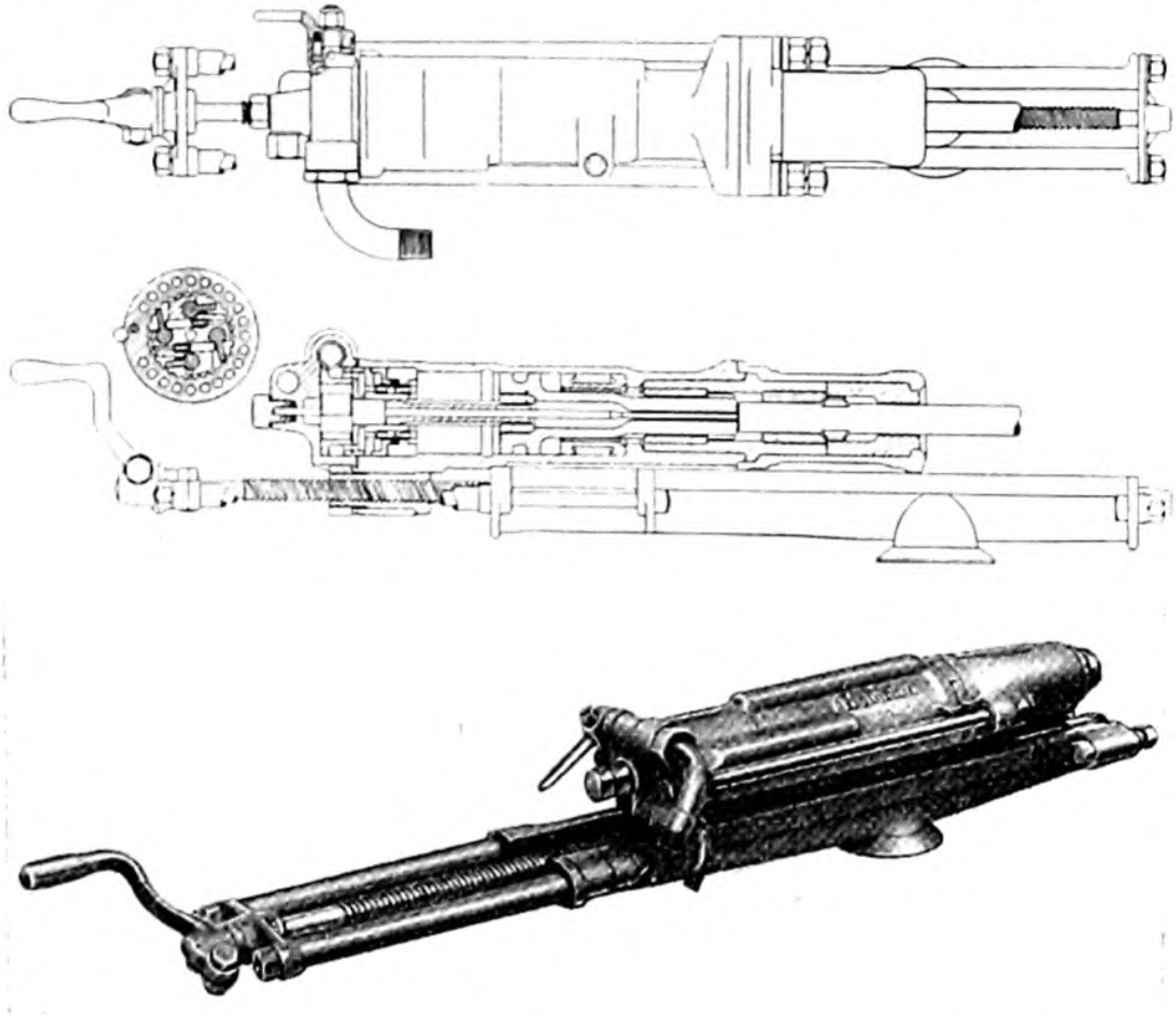


Fig. 107

on the cradle, giving greater flexibility in drilling and setting up. The cradle is equipped with a hand-operated feed screw to provide the forward feed to the machine and to maintain alignment. The machines are usually arranged for wet drilling, water being supplied to the bit through hollow drill rods. The automatic feed on some machines is provided by a motor, integral with the machine as in the stoper type, or mounted at the back end of the cradle and attached to the feed screw, as shown in Figs. 109A and B. Mounting the automatic feed device at the back end of the cradle eliminates any direct shock from the running drill. In both the hand or automatic feed machine the feeding device allows the advance or retraction of the

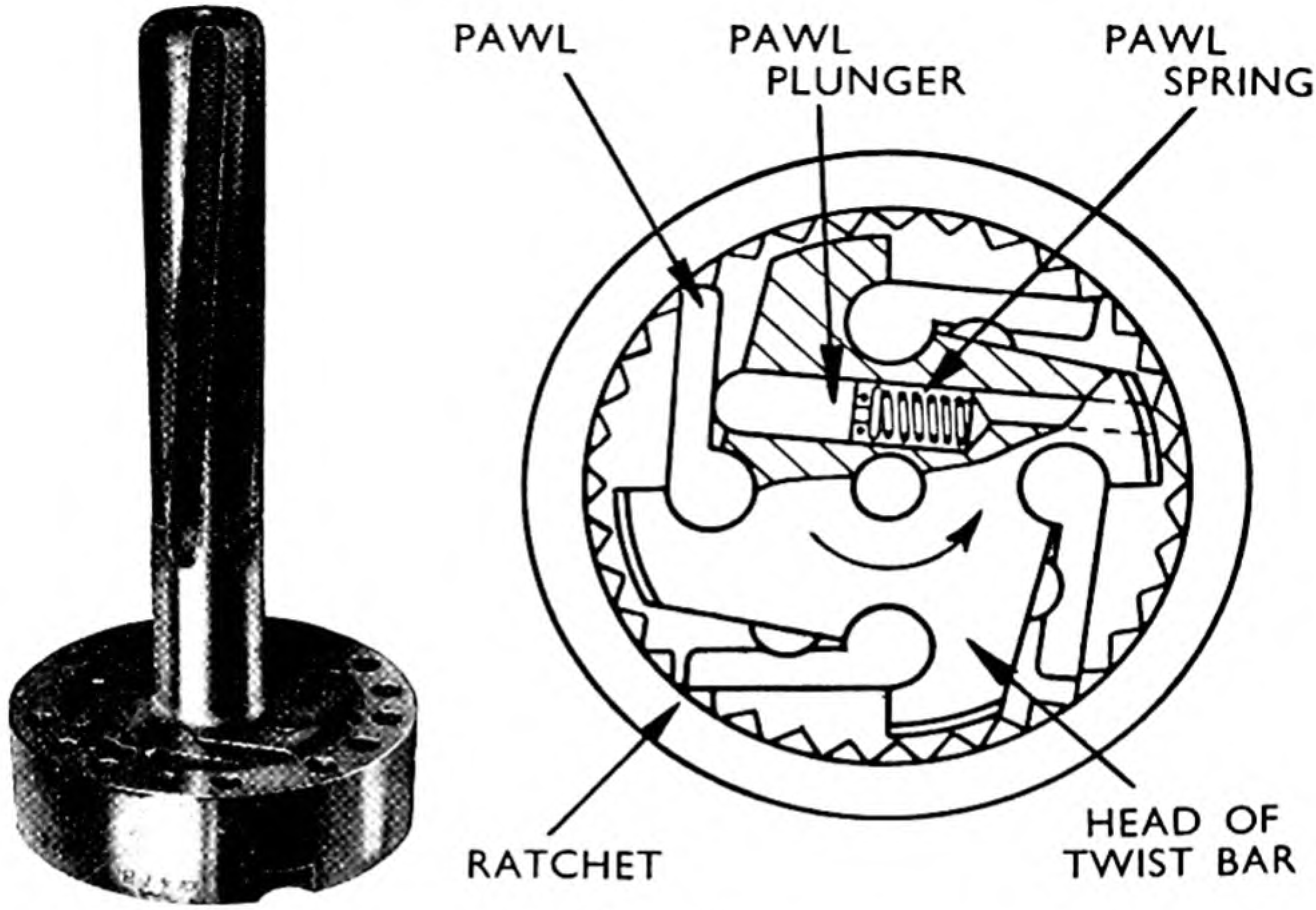


Fig. 108

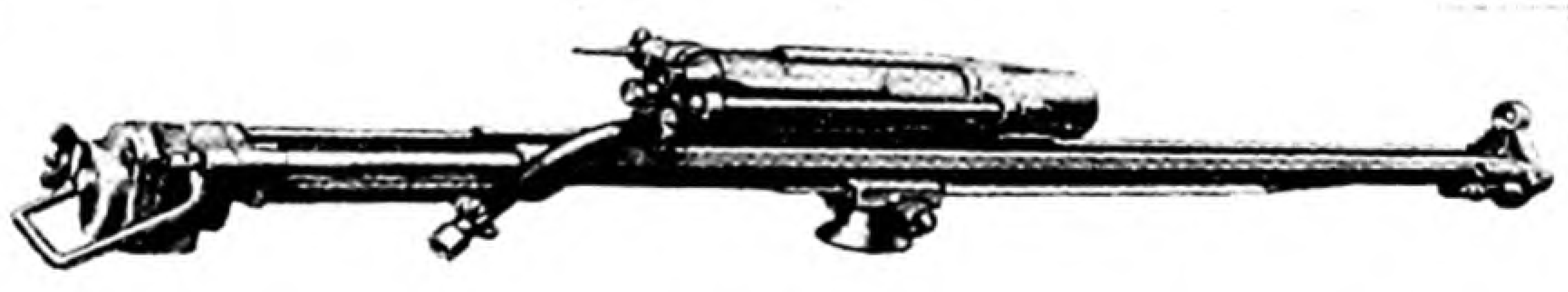


Fig. 109A

Fig. 109B

drill steel as required. The automatic feed machine gives a faster drilling time and quicker drill steel changes. With the advent of the carbide bit where fewer drill steel changes are necessary, this type of

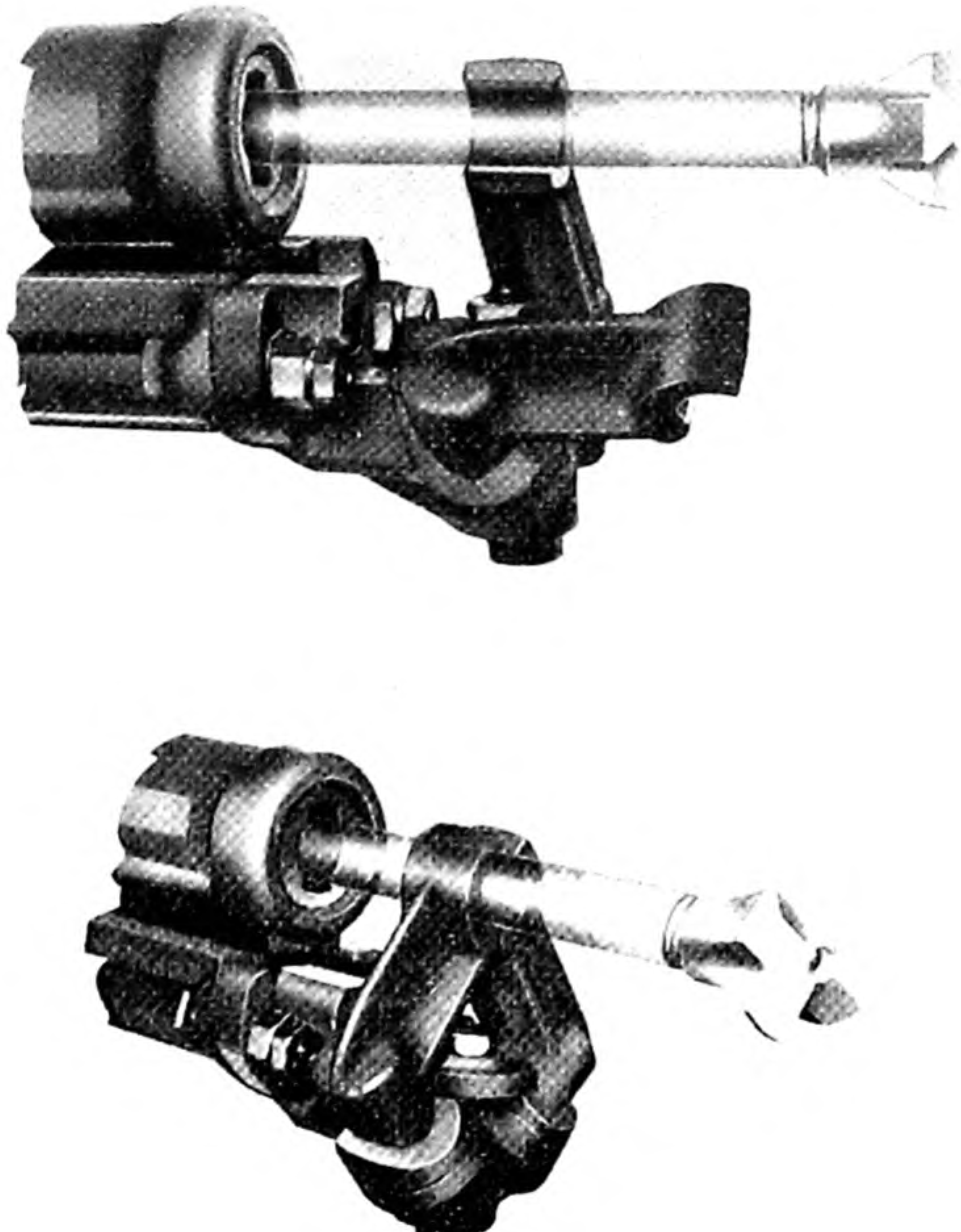


FIG. 110

machine has a great advantage. The front end of the machine cradle is usually provided with drill-steel centralisers or aligners which can be sprung back when not in use, *vide* Fig. 110. With a suitable mounting the drifter can drill holes at any desired angle, and the machines work normally with compressed air at a pressure of from

50 to 80 lb. per square inch. The following table gives the average sizes and weight of drifters in general use:

Bore	*Weight*	*Use*
inches	*lb.*	
$2\frac{3}{4}$–3	100–130	Light duties.
3 –$3\frac{1}{2}$	130–160	Medium duties.
$3\frac{1}{2}$	160–200	Heavy duties.

At a pressure of 80 lb. per square inch the air consumption varies from approximately 125 to 165 cubic feet per minute and the blows from 1,600 to 2,100 per minute. The usual feed length is 24 inches,

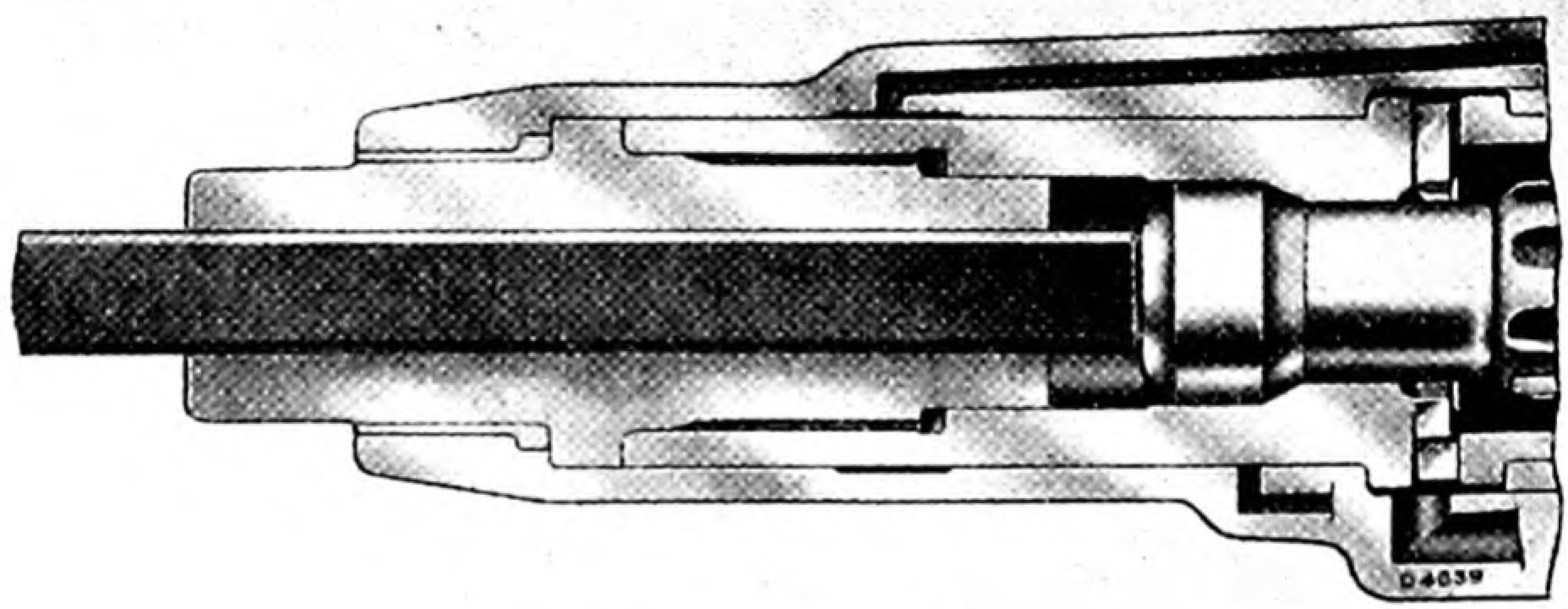


FIG. 111

but with some heavy machines this may be from 30 to 40 inches. The chuck construction varies according to the type of drill steels used, which may be lugged steel, hexagonal or quarter-octagon steel.

The anvil block chuck, Fig. 111, in which the piston strikes an anvil block interposed between the piston and the drill steel, is used exclusively on the stoper type and is strongly favoured for most types of drifter. The use of the anvil block eliminates the necessity to form a shank on the drill steel but introduces an additional loss of power in the blow which is not found with the open-chuck type. Improvements in machine design have reduced this loss in the effective piston blow, and the anvil block chuck is in extensive use. Drills using collared drill steel incorporate a steel retainer consisting of a spring-supported yoke which clips over the

drill steel in front of the collar and which can be swung back to release the drill steel. Fig. 112 illustrates a drifter utilising collared drill steel. The lugged drill-steel chuck is designed to accommodate a drill steel on which a lugged shank has been forged, and these are shown in Fig. 113A and B. The hole in the chuck key in the nose of

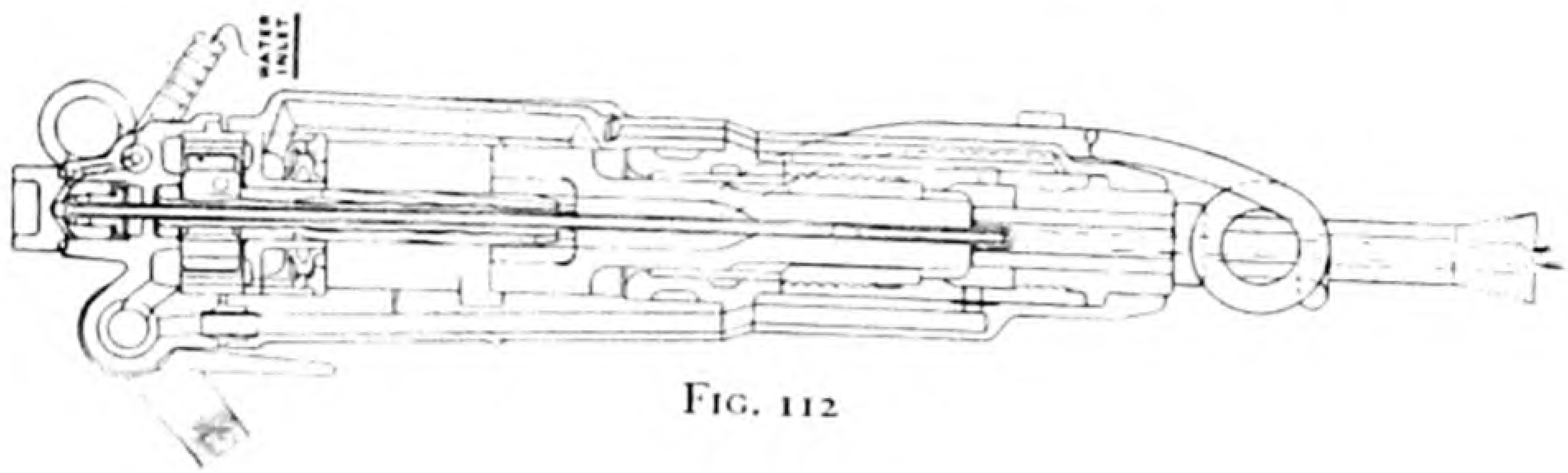


FIG. 112

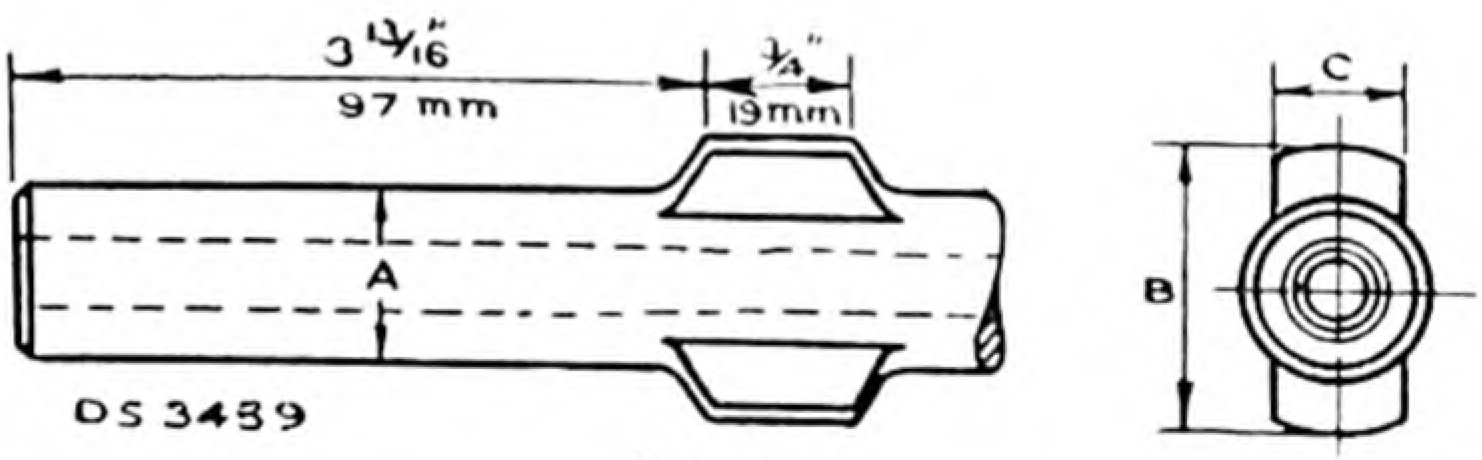


FIG. 113A

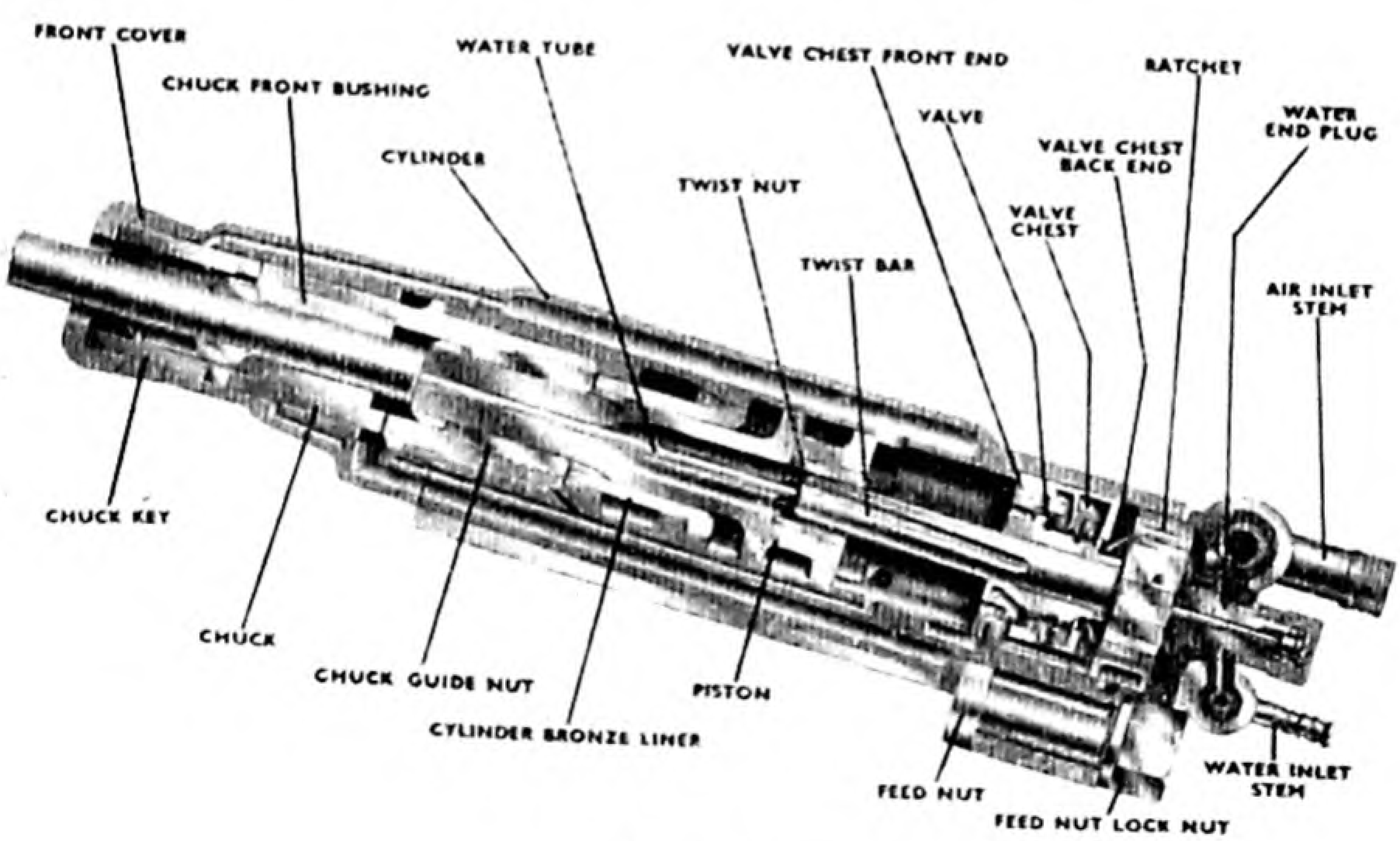


FIG. 113B

the drifter permits the lugs to pass into a recess formed at the rear of the key. A bush behind the recess limits the drill steel entry to the correct striking point, while rotation of the steel is transmitted through the lugs.

Various modifications in the design of the drifters allow either exhaust or live air blowing to clear the hole of sludge or chippings.

Hand-held hammer drills. This type of drill is equipped with handles and has a construction similar to that of the drifter but is much lighter. Various modifications are in use to convert these machines to the mounted type or to enable the operator to have a better control of the machine. The following table gives the classes of drills in general use:

Bore	*Weight*	*Use*
inches	*lb.*	
$2\frac{1}{2}$	25–35	Coal and soft rock.
$2\frac{1}{2}$–$2\frac{5}{8}$	35–50	Medium and general purpose.
$2\frac{1}{2}$–3	50–60	General purpose.
3–$3\frac{1}{2}$	110–130	Sinkers.

The sinker, or heavier hand-held drill, is a drifter equipped with handles for heavy work in shaft sinking.

The hand-held drill is normally used for downhole work, but is often applied to horizontal and inclined holes. Where the conditions allow, a hand-held drill may be mounted on a cradle as in the case of the drifter. These mounts are provided with flanges to slide in a feed-screw cradle, which is normally of the sliding-cone type and capable of being attached to a drill clamp for fixing the machine to the horizontal arm of a drilling column. The 'air-leg' mounting incorporates a fork to which the hand drill is fixed, the fork being held at the top of the inner operating cylinder of the leg. The outer cylinder is provided with a claw at the base against which the drill operates, as shown in Fig. 114. The piston operation of the air-leg can be controlled independently from the drilling machine, and by adjustment of the leg the drill can be operated at any height within the reach of the leg.

The hand-held hammer drill is generally employed on the Continent, in which case it is column mounted and used for drifting and cross-cutting. In Britain this drill is used when drilling in medium

and soft measures, and for a variety of other tasks. Drill-steel chucks are made for collared, lugged or straight drill steels for use with anvil block machines. Several designs of vented-head machines are in use and blowing can be carried out by exhaust or throttle control. The machines are designed for 'wet' or 'dry' operations as required.

Stoper hammer drill. The stoper is a hammer drill in which the machine is in line with the air-operated feed cylinder. The construction of the drill is similar to other hammer drills and in present-day

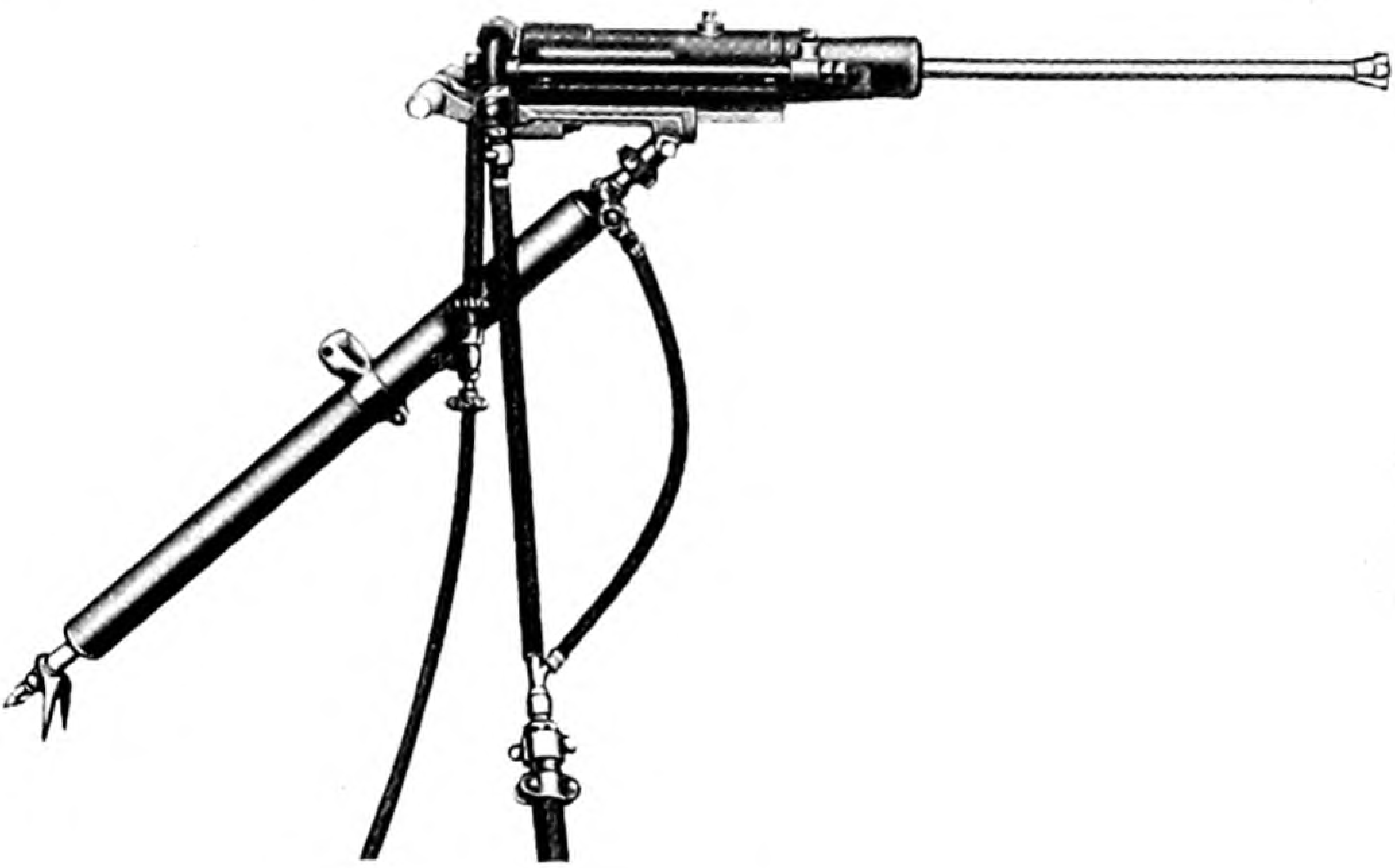


FIG. 114

practice usually has automatic rotation. The chuck is always of the anvil block type accommodating a straight drill steel. The piston cylinder is provided with a steel point to support and maintain the machine and drill-steel alignment. This type of machine is used principally for top holes, and as such is used in stope drilling in the metalliferous field.

(*i*) *Drilling rigs and carriages.* Where high drifting speeds are not essential, a simple rig incorporating the normal stretcher column may be all that is required, since it is not economical to install expensive high-capacity equipment for drift driving unless the

equipment can be worked at or near its full capacity. Where drifts are being driven for the normal development of new areas and this work is well in advance of extraction, there is no necessity to drive at a high rate of advance. In some instances, however, there is a need for fully mechanised drifting and high rates of advance, in which case the reduction of drilling time to a minimum, along with other factors involved, becomes important in order to attain the required rate of advance.

It is estimated that the actual drilling operation takes approximately from 25 to 30 per cent. of the total available time, and the effect of saving several minutes in drilling means a considerable saving in manshifts per yard. Macaskill and Evans * give the following figures:

Saving possible by a reduction of 20 minutes in drilling, loading or support-setting times.

Equipment	*Possible increase in yards per week.*		*Saving in manshifts per yard.*
	From	*To*	
Eimco into 15-cwt. tubs, 2 air-legs	17·5	18·0	0·1
8 B.U./4 into 15-cwt. tubs, 3 air-legs	20·2	21·0	0·2
11 B.U. on to belt, 3 drill rigs	26·5	28·9	0·3

The use of the *air-leg* has already been referred to. This form of drill mounting does effect a saving in man-power when compared with drifts in which hand-held machines are being used, although the reduction in overall drilling time is not much less under similar conditions.

The use of *drill rigs* or *jumbos* mounting two or more drifters has appreciably affected the organisation and technique of drift drivage. The normal drilling carriage (or German gadding car) is, in fact, a carriage running on a track, to which is attached a tubular framework of horizontal and vertical arms coupled by suitable clamps and capable of mounting two, three, four or more drifters. The carriage is provided with track clamps. This form of carriage has been criticised due to the time taken to set up the drills, since 'spanner work' is considerable.

* *Proc. National Assoc. Coll. Managers* 1949–50.

The German *Bohrwagen*, Fig. 115, is a tubular steel frame mounted on a four-wheeled carriage, usually running on the two outer rails of the haulage track. The frame can be jacked to the roof at AA by air cylinders and, if necessary, clamped to the rails. The frame is hinged at BB so that the frame can be swung into the horizontal position for travelling or shot-firing. The main disadvantage of this type of rig is that, in order to clear the frame, the loading equipment is limited to a low-built machine which is usually a 'shovel loader' of the German type. The drilling rig must be removed prior to shot-firing, such movement requiring the horizontal members to be swung into line with the drift.

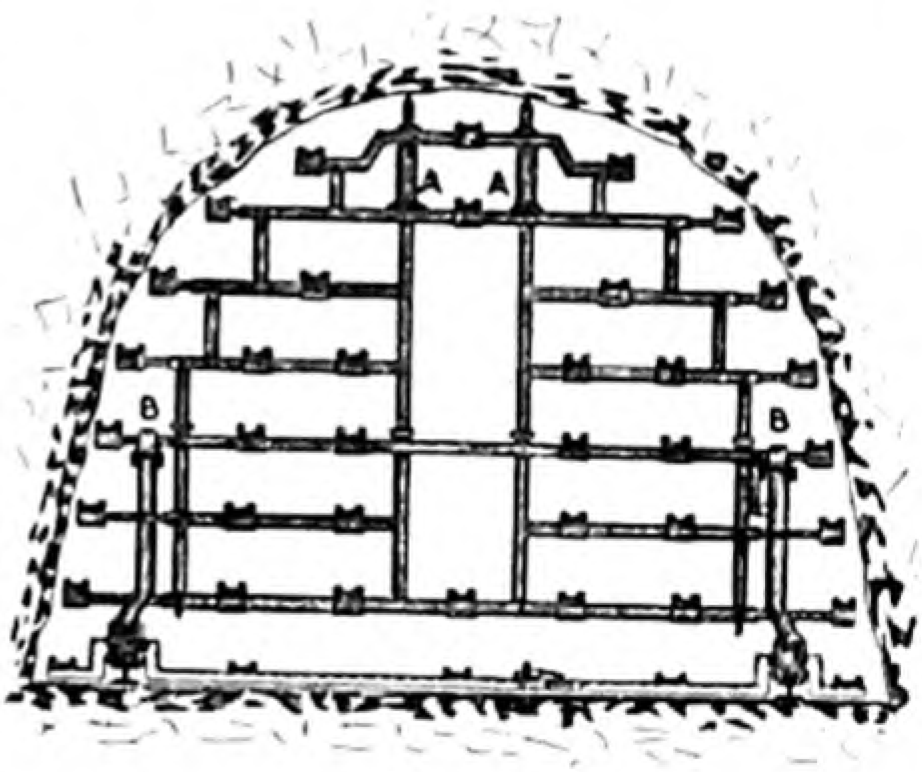


FIG. 115

The Ingersoll-Rand air-operated D.J.M. jumbo is essentially a drilling carriage on which two columns are mounted, each column being capable of swinging from the horizontal carrying position to the vertical operating position and being able to be extended pneumatically between roof and floor to give a rigid support to the drill mounting, as shown in Fig. 117. The carriage is fitted with air and water manifolds. The columns are $3\frac{1}{2}$ inches diameter and have a minimum length of 8 feet 4 inches and are extensible to about 11 feet. The swivel centre for a column is 12 inches from the centre line of the carriage, while the column has a swing of 22 inches which, with a 30-inch arm, gives a maximum spread of 64 inches from the carriage centre line. Each vertical column takes a 30-inch arm which can be rotated through 360 degrees and to which a drifter can be mounted. A rig of more recent design is illustrated in Fig. 118. The design of the jumbo allows easy handling and quick positioning of the drifters, while the carriage is provided with accommodation for three sets of drill rods, bits, hose connections, spanners and other accessories.

The jib mounting (Joy-Sullivan), illustrated in Fig. 119, enables the drill, which is mounted at the end of the jib, to be moved to and fixed in any position hydraulically. The joints at the end of the jib enable drilling to be done at any angle with the cradle over- or

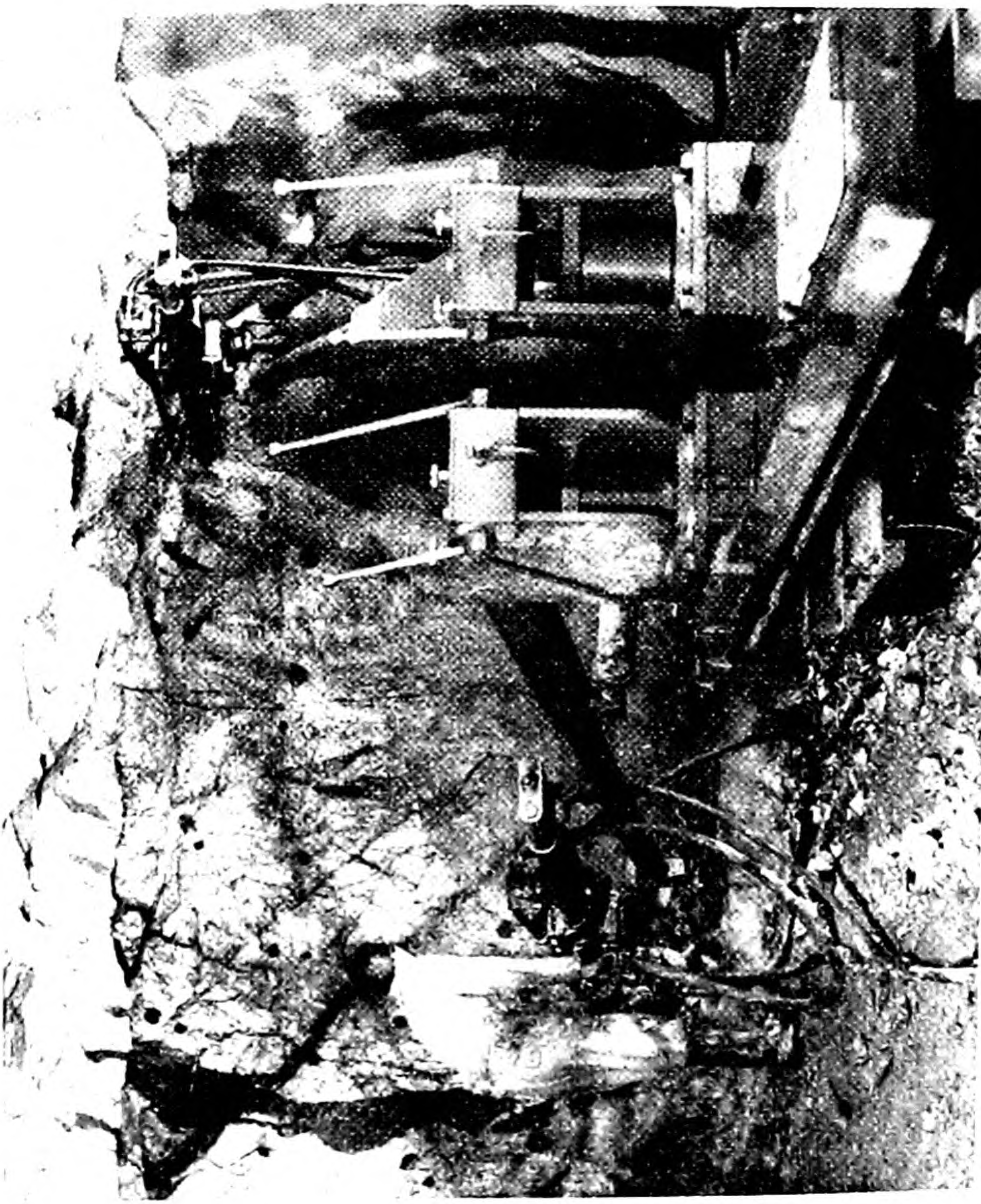


FIG. 116.—The Holman drilling rig incorporates a double jib hydraulic mounting for two drifters as illustrated. Each boom is independently operated and gives a wide range to the two drifters over the face being drilled.

under-slung, and the drill feed can also be controlled by hydraulic power. The carriage, when mounted on rails, can be hydraulically jacked to the roof, and hydraulic braking ensures extra rigidity. This drilling rig incorporates flexibility with the capability of covering any normal drift size. The hydraulic pump may be either hand-

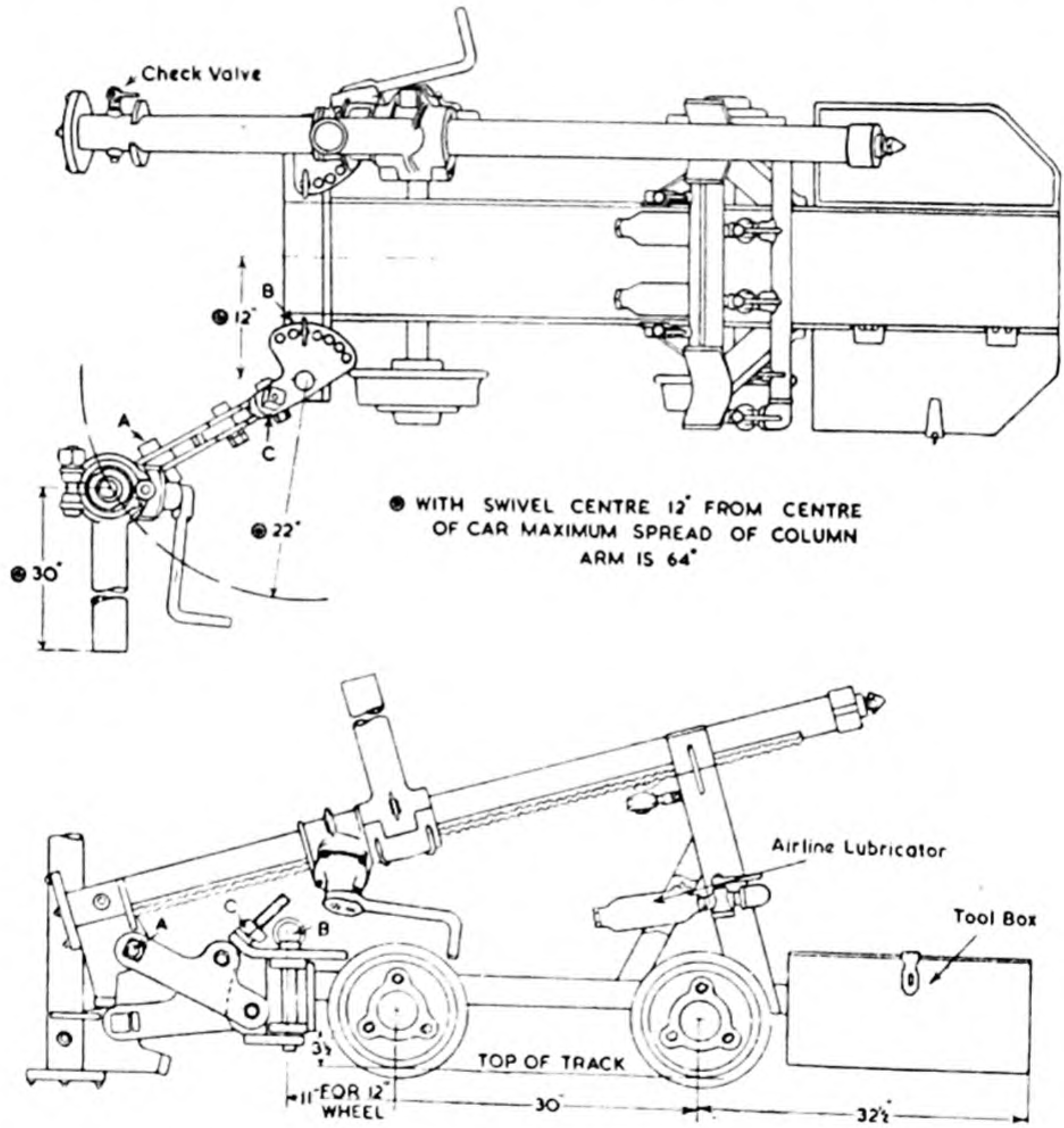


Fig. 117

operated or driven electrically. In the latter case, several jibs may be operated by remote control from the pump. With the hydraulic pump control it is possible to elevate the jib and drill to any position within 45 seconds. Lowering of the jib is controlled by a small valve lever. A two-drill rig is shown in Fig. 119. This unit has also been adapted to a Joy T.2 truck mounting, providing a trackless machine.

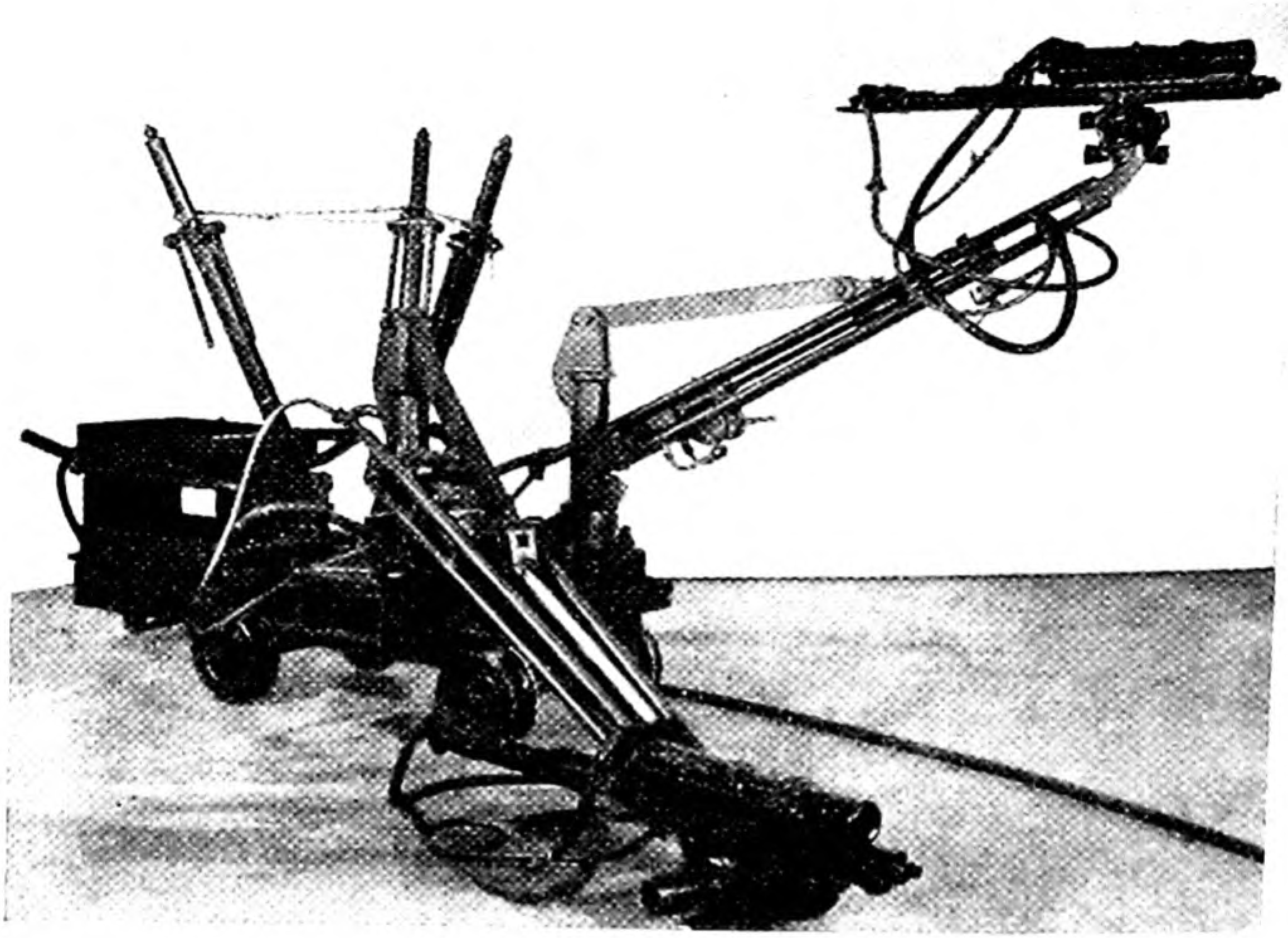


FIG. 118

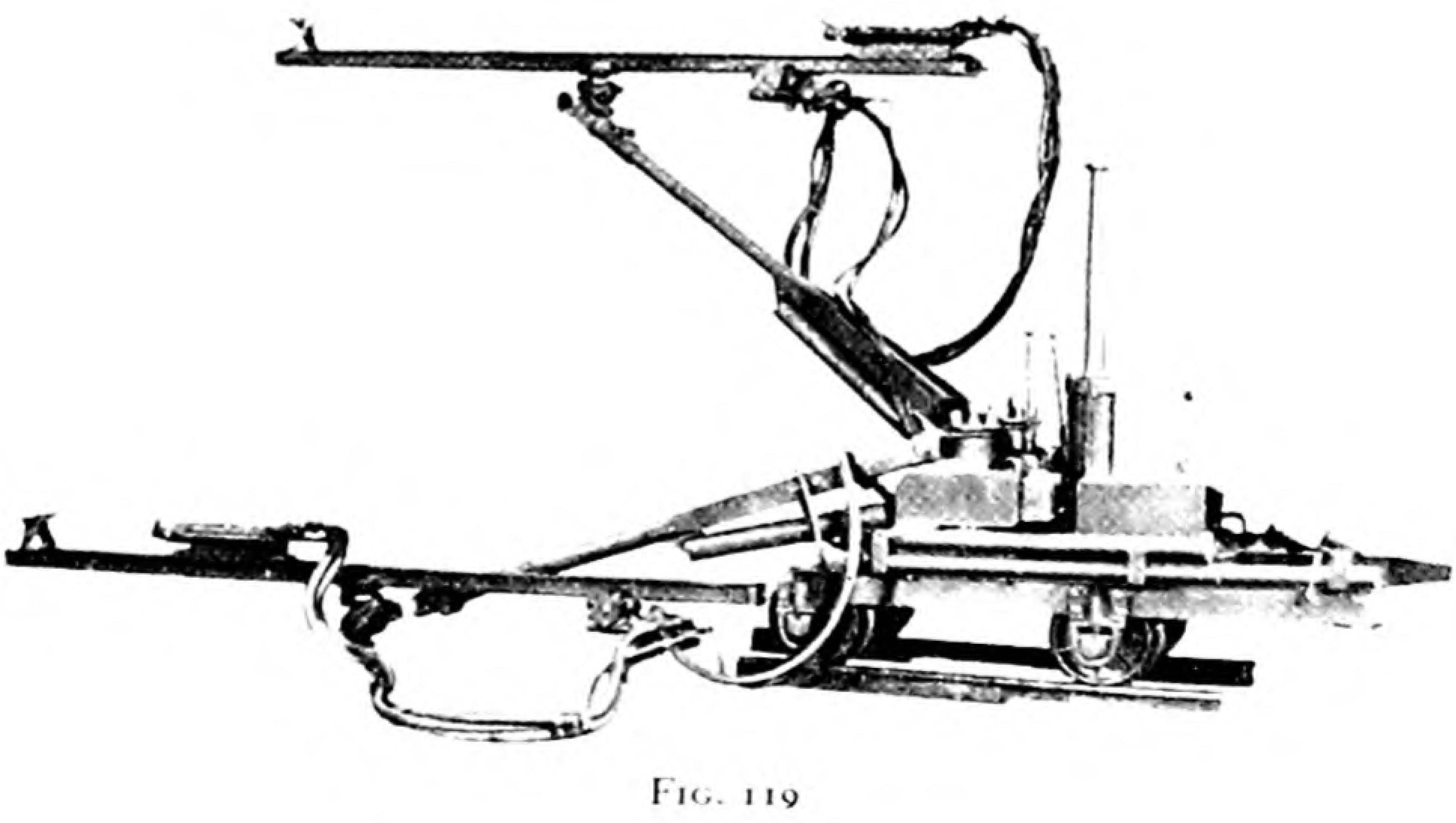


FIG. 119

The truck with a two-drill mounting may be cut down in size to enable it to be accommodated in drifts less than 12 feet in width. For wider drifts the truck mounting is retained at the original width of about 6 feet 6 inches to accommodate three drifters, as shown in Fig. 120. The truck-mounted three-drill rig is shown in operation in Fig. 121. This unit is an example of the fully mechanised drilling equipment in use in high-speed drifting projects. It takes full advantage of the adoption of carbide bits, gives a longer feed with fewer steel changes, and so reduces the overall drilling time.

The following time-study * is illustrative of the capabilities of the unit in average coal-measure strata:

Joy Sullivan three-drill rig—Yorkshire colliery

Excavation . .	14 ft. × 10 ft.
Rock . . .	Blue Shale or Bind.
Drilling Machines .	Holman S.L. 200.
Drilling Bits . .	$1\frac{11}{16}$-in. carbide bits Holbits.
Personnel . .	4 men, one man on each drill and one man on drill rig operating hydraulic controls and slackening swivel nut.

	Left-hand Machine	*Centre Machine*	*Right-hand Machine*
Number of Holes . .	11	10	11
Total Length . .	71 ft. 3 in.	67 ft. 9 in.	72 ft. 11 in.
Total Operational Time .	35 mins. 16 secs. 100%	30 mins. 26 secs. 100%	35 mins. 37 secs. 100%
Drilling Time . .	24 mins. 16 secs. 68·8%	23 mins. 40 secs. 77·8%	25 mins. 48 secs. 72·4%
Supplementary Time .	10 mins. 59 secs. 31·2%	6 mins. 46 secs. 22·2%	9 mins. 49 secs. 27·6%
Av. Drilling Time per Hole	2 mins. 12·3 secs.	2 mins. 22 secs.	2 mins. 20·7 secs.
Av. Supplementary Time	59·8 secs.	40·6 secs.	53·5 secs.
Penetration Speed . .	2·94 ft. per min.	2·87 ft. per min.	3·15 ft. per min.

Total No. of Holes .	32	1st machine started at 10.10 a.m. and last machine finished at 11.03 a.m.
Total Length . .	211 ft. 11 in.	
Total Operational Time .	53 mins.	
Av. Operational Time per Hole. . . .	1 min. 41 secs.	Supplementary time is inclusive of positioning, changing bits, altering and replacing slide.
Av. Supplementary Time per Hole . . .	49·6 secs.	

* *King's College Mining Bulletin*, No. 5, Series Mech. No. 25—Drifting.

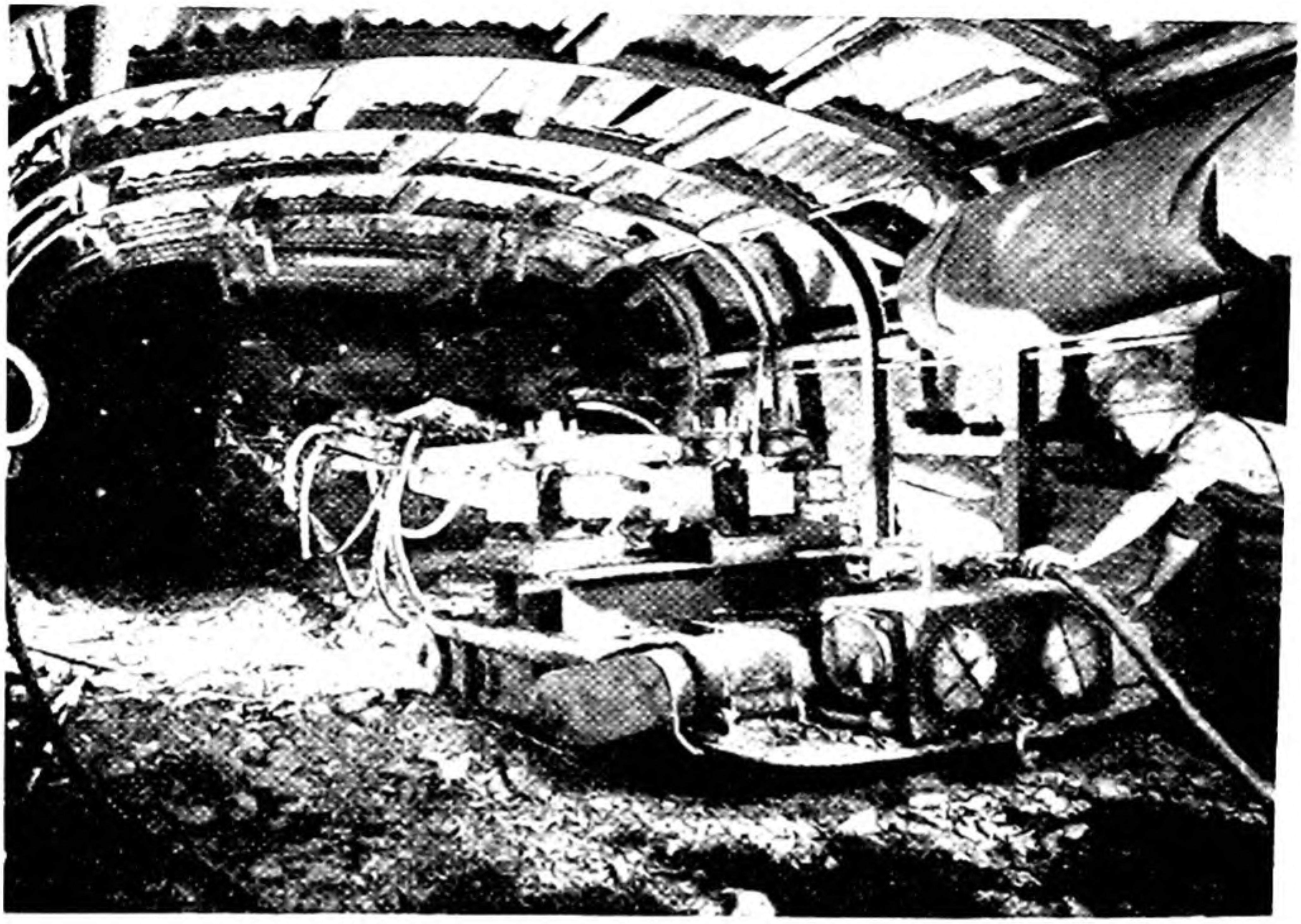


FIG. 120

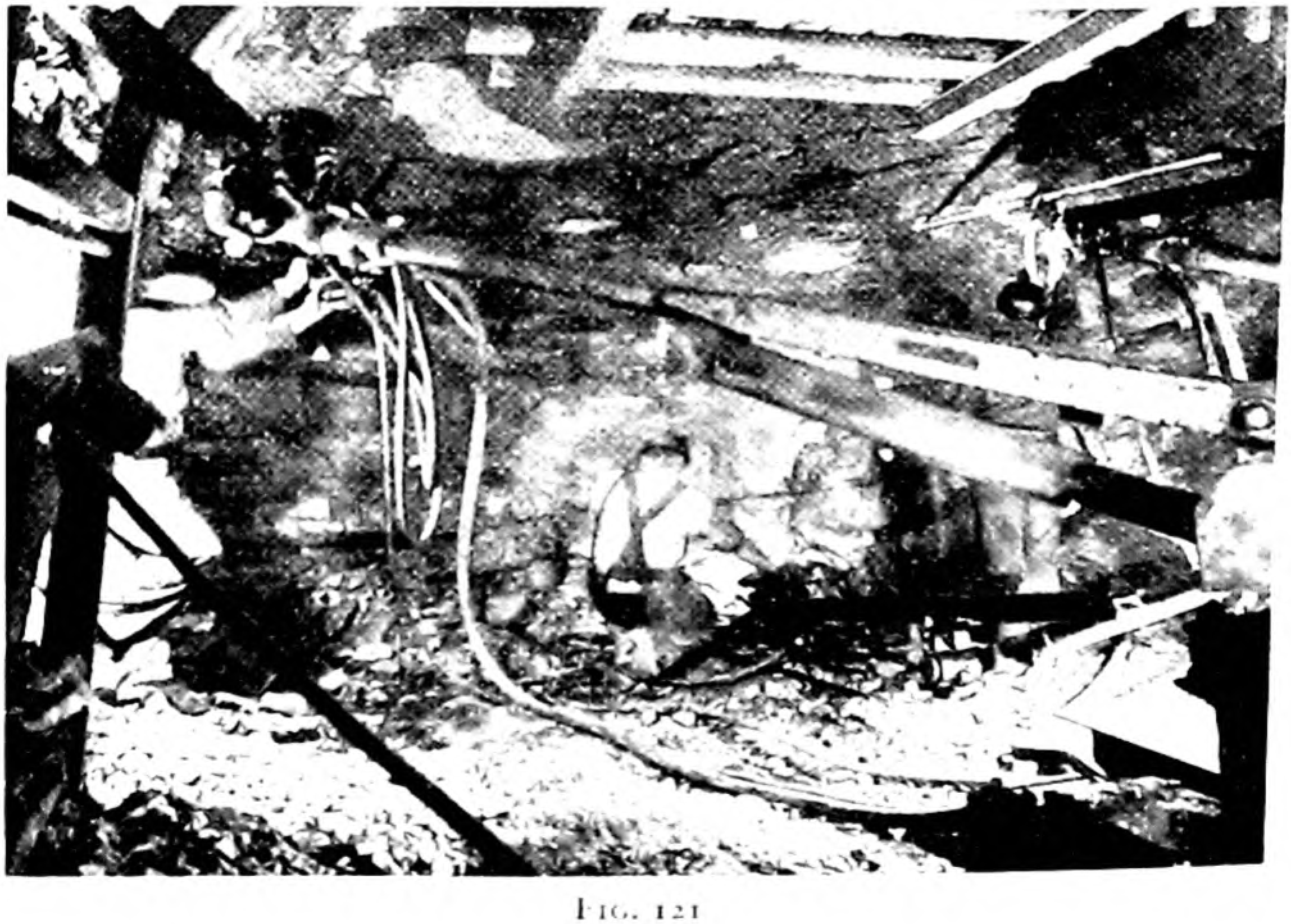


FIG. 121

(*j*) *Removal of the dust from the shot hole and dust suppression.* To facilitate the removal of the dust from the shot hole, twisted drill rods are used in dry drilling in the case of horizontal or slightly inclined holes. Since in percussive drilling the drill steel rotates at from 100 to 200 r.p.m., it is easy to convey the dust from the back to the front of the hole. Such rods are also used with rotary drilling machines, where the rotative speed may be from 500 to 600 r.p.m. down to 150 to 160 r.p.m. When drilling upwards, the dust is removed automatically, and it is possible in this case, as in the case of wet drilling, to use round and hexagonal steel rods.

In the case of holes drilled downwards, either in a vertical or steeply inclined direction, it is possible to blow out the hole by using hollow rods, but this produces heavy dust formation. Similarly, the shot hole can be filled with water and, using twisted drill rods, it is possible to drill from 3 to 5 feet without removing the drill rod from the hole. Dry drilling has a decided disadvantage from the point of view of removing and preventing the formation of dust. With percussive drilling, the dust formation is heavy, due to the pulverising action of the drill. Rotary drilling machines do not produce so much dust, and the dust is coarser in size and, therefore, less dangerous.

The danger from silicosis depends upon the size and kind of dust which is being inhaled. The dust from calcareous sandstone and sandstone marls is harmless, while the dust produced from rocks containing free silica (quartz) or certain silica compounds (sericite) is particularly dangerous. Sandstones, sandy-shales and most of the clay slates of the carboniferous series belong to this group. The most dangerous dust is that which is so fine that it remains air-borne in a fine dust cloud for a considerable time. This dust is regarded as the main factor in the incidence of silicosis among miners, the dangerous particle size being under 5μ.

The most effective means of combating the dust hazard in rock drifting is by wet drilling. This method can be applied in the three following ways:

(i) Using solid drill rods and a small mist spray at the mouth of the hole.

(ii) Using twist drills and water or foam equipment, with a small pipe placed into the hole and from which the water or foam is directed into the hole. A water spray can also be used at the front of the hole.

(iii) Using hollow drill rods with a central water-supply and with or without a flushing head.

The most effective method of dust suppression is to use hollow drill rods with a flushing device on the drill. The flushing head on the drilling machine allows water or foam to be circulated to the bottom of the hole so that the dust is attacked at the point of formation. The dust is removed as a mud, either by using twist drill rods or fairly high-pressure water. Another advantage of the method is that the bottom of the hole is kept clean and allows the bit to give a

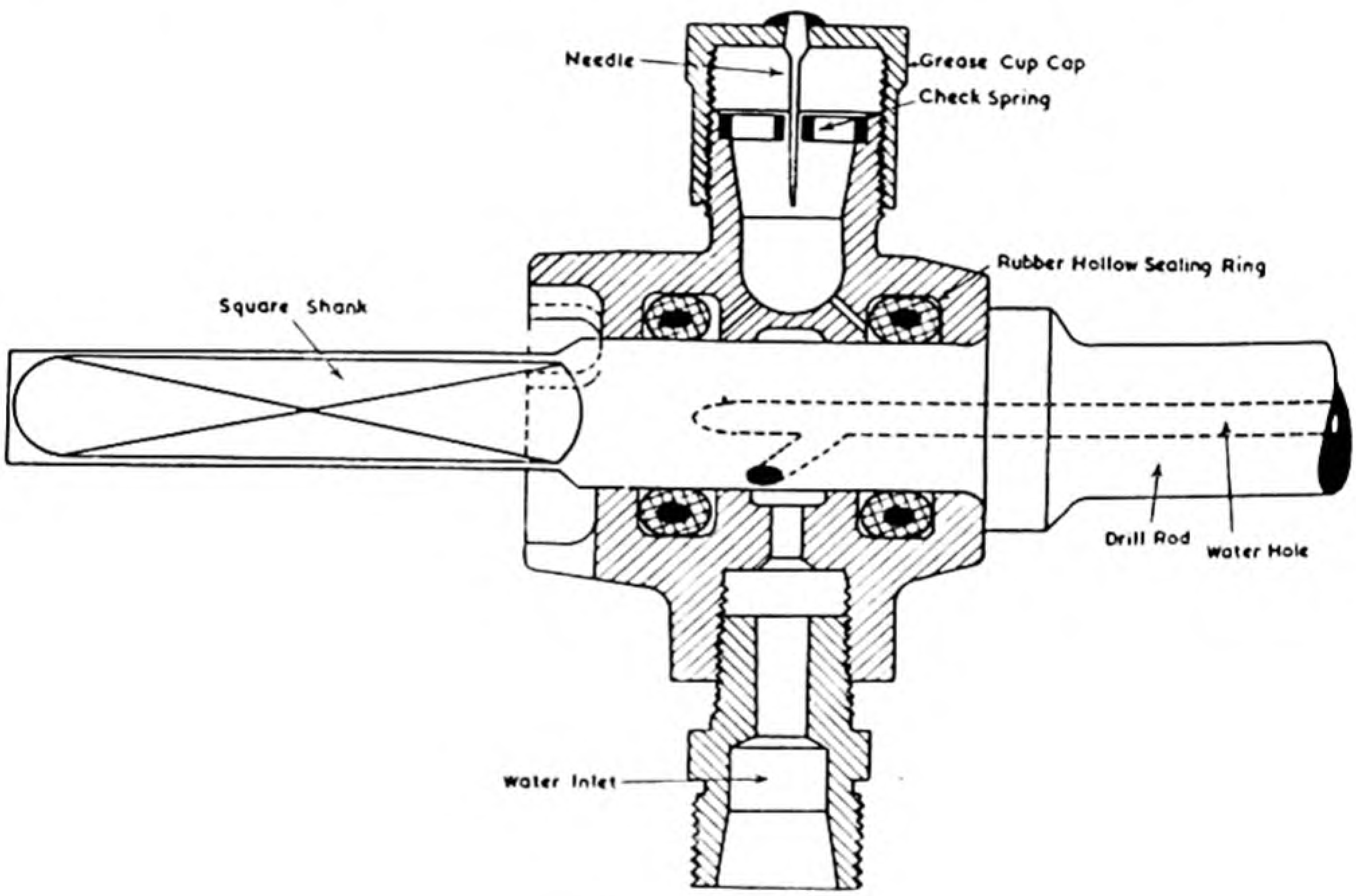


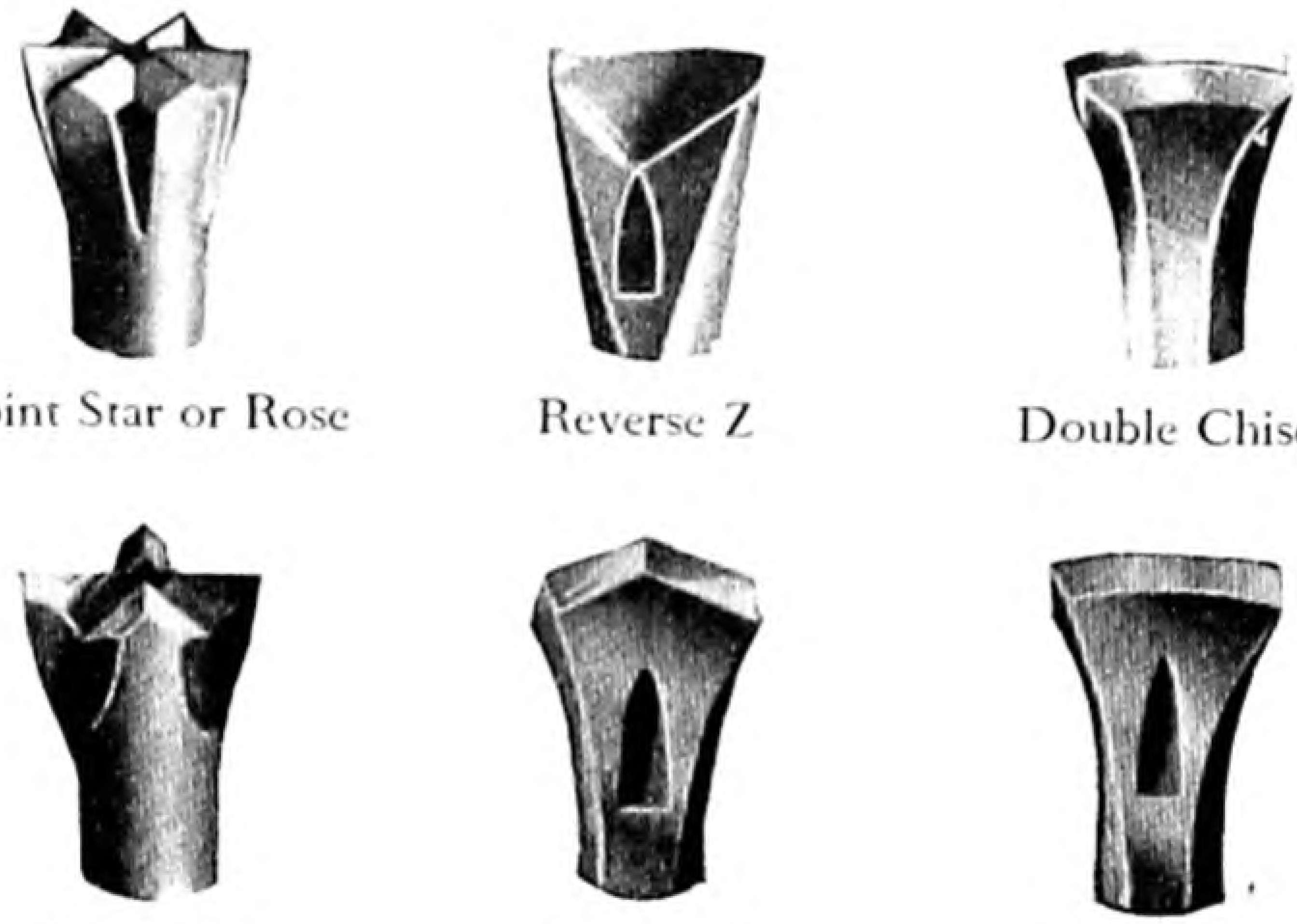
FIG. 122

better performance; it also reduces excessive abrasive wear on the bit, and the drilling efficiency may be from 5 to 20 per cent. higher than in dry drilling. Fig. 122 is an example of the water inlet or flushing device on the drilling machine. The flushing water can come either from a water main or from a portable pressure tank in which the water is ejected at the required pressure by a suitable coupling to the compressed-air main. The tank (at least 200 gallons) is usually large enough to operate four drills.

(*k*) *Drilling rods and drilling bits.* Hollow or solid drill rods are used. Hollow drill rods may have a round or hexagonal section and

twist steel has been introduced. With solid drill rods, hexagon and twisted rods are in use. The rods or drill steels are made in a range from $\frac{7}{8}$- to $1\frac{1}{4}$-inch diameter, depending upon the diameter of the bit and explosive cartridge used.

Because of the arduous duty of the drill rods, which have to withstand heavy blows and vibration, only good-quality steel can be used. Crucible steel, free from flaws, is normally used, with a carbon content from 0·7 to 0·8 per cent. Where the rock is hard and abrasive, higher carbon steel up to 1·1 per cent. may be used, and in such cases the drilling bits may be either integral with the rod or detachable. In



6 Point Star or Rose — Reverse Z — Double Chisel

4 Point Cross — Diamond — Single Chise

Fig. 123

the former case the cutting edge of the bit is forged on to the drill steel, using forging and forming dies, and Fig. 123 shows a series of common shapes.

The simple chisel bit can be made easily and has been used frequently in compact, unbroken ground. Since it has little lateral guide, a symmetrical round hole is not obtained and the double chisel bit with a double cutting edge is preferred, *vide* Fig. 123. With this type of bit the presence of cracks in the strata causes no difficulty. With hard strata the lateral wear is excessive, and the reduction in the bit diameter results in frequent changes in bore requiring a wide range of bit diameters. The cross-bit type obviates this difficulty.

The 'Z' bit, which has a simple cutting edge and two lateral edges, is only suitable for a mild and intermediate hard rock.

The cross-bit can be used for intermediate strata and is well suited for broken ground and deep drilling, and has therefore a greater range of usefulness. The reduction in diameter by lateral wear is very small, requiring a smaller range of bit diameters. The graduation in diameter of bit need not be more than $\frac{1}{16}$ inch, so that the initial diameter can be smaller and the drilling efficiency consequently higher.

The 'X' bit is used in some strata where the normal cross-bit tends to give a pentagonal hole or where the ground breaks in such a way that chippings are liable to stick between the wings of the bit.

The bits with multi-wing construction and crown bits must be notched sufficiently between the cutting edges to allow the chippings to get away freely in the water stream. They are suitable for extremely hard and tough strata where another type of bit would suffer excessive loss in diameter.

The final bit diameter to be chosen depends upon the diameter of the explosive being used, which is normally $1\frac{1}{4}$ or $1\frac{7}{16}$ inches. The diameter of the drill hole should be about $\frac{1}{4}$ inch greater than the explosive cartridge. The size of the bit is limited by the sectional diameter of the drill steel. The smallest bit diameter is about $\frac{1}{4}$ inch more than the diameter of the circumscribed circle on the drill steel section. The largest bit diameter should not exceed two and a half times the diameter of the drill steel, while with crown bits twice the bit diameter should be the maximum.

Detachable hardened steel bits have been introduced for percussive drilling in order to eliminate the transport of drill steels to the surface for reforging and sharpening and the difficulties arising from the constant heating and forging of the drill steels. The 'PM' or Padley and Morgan bit, illustrated in Fig. 124, is one of this type evolved on the Rand, resulting in a reduction in the consumption of steel for drill rods. The cutting edges of these detachable bits have taken the shape of the four-point cross, the single chisel and the 'Z' and reverse 'Z' types. The PM bit, which is primarily designed for a single tour of duty and termed a 'one pass' bit, is forged in one operation and in use is run to destruction. The efficiency of blow transmission is fairly high, the bit having a female tapered socket fitting into a male counterpart on the rod. This type of connection would appear to be ideal when the bit is used to destruction and not resharpened, thus requiring to be bedded only once into the rod.

Another form of detachable steel bit, shown in Fig. 125, incorporates an alloy steel stud connection to the rod, the stud having been forged into the bit after the threads have been cut. The stud type affords a larger bearing surface between the bit and the rod, giving

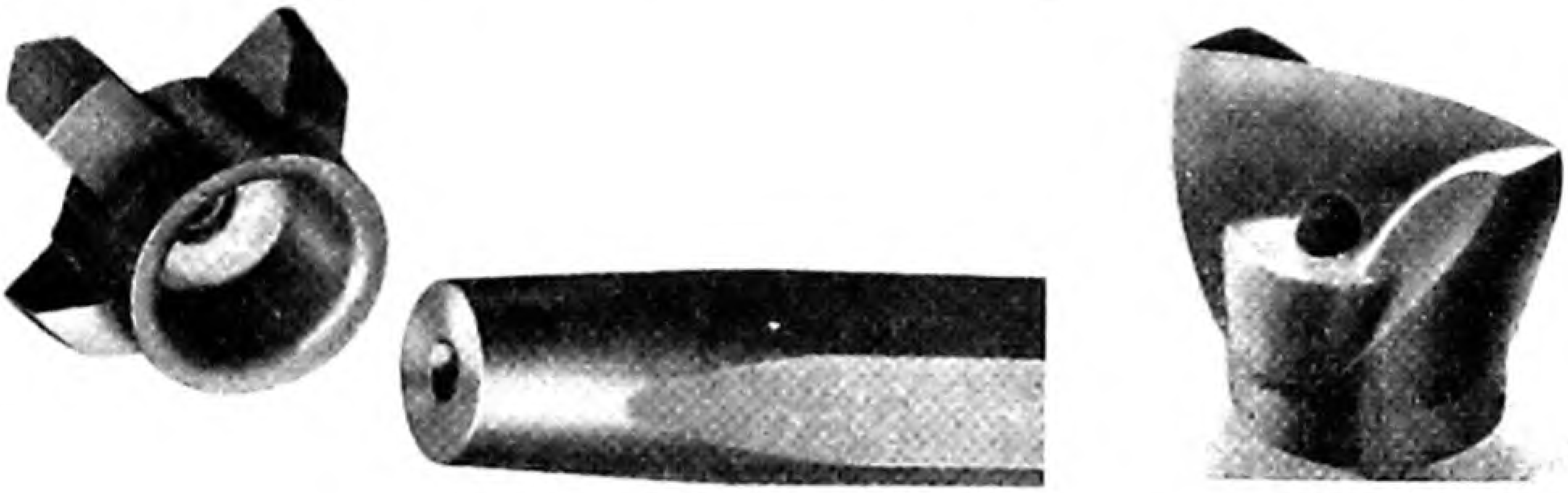


FIG. 124

FIG. 125

a maximum power transmission to the cutting edges. The 'Rip Bit', shown in Fig. 125, is resharpened by grinding or hot milling. These bits are usually provided in $\frac{1}{8}$-inch changes from $1\frac{3}{8}$ to $2\frac{1}{2}$ inches with a $\frac{3}{4}$-inch stud, or from 2 inches to $3\frac{1}{2}$ inches with a 1-inch stud for heavy drilling.

The change from mine steels to detachable bits has resulted in economy in transport and maintenance cost and increase in drilling

efficiency. The throw-away type of bit is applicable to severe drilling conditions, while the reconditionable hard-steel bit will be used where the rock is fairly soft and non-abrasive. The deciding factor will be one of economics, the initial cost of the throw-away type being one extreme and that of the tungsten carbide or alloy-tipped bit being the other, with the detachable steel bit occupying a middle path.

(*l*) *Cemented carbide-tipped bits.* Instead of hardened steel bits, bits incorporating cemented carbide cutting tips are being introduced extensively for drilling both stone and coal. Several examples are illustrated in Figs. 126A and 126B.

Cemented carbides are mainly carbides of high melting-point

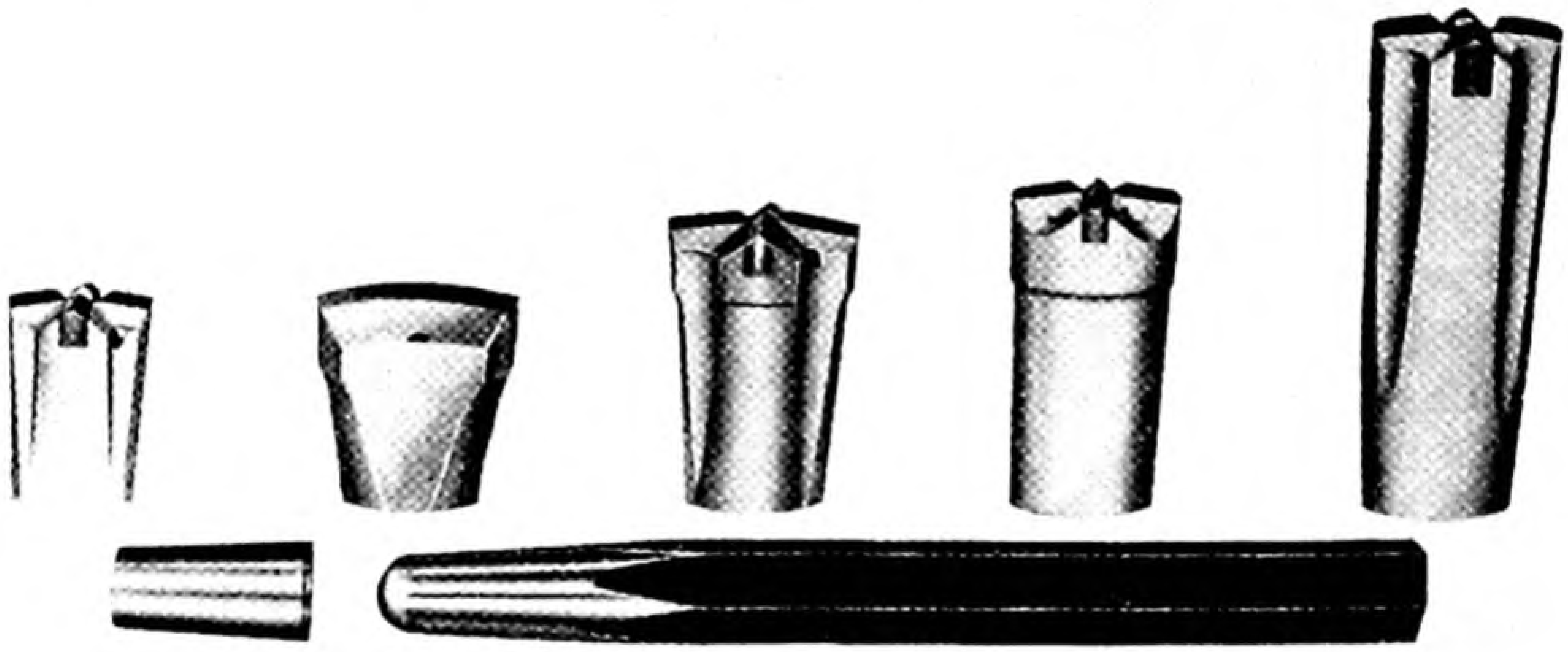


FIG. 126A

metals, such as tungsten, having a specific gravity of about 17 and a Mohr hardness of 9·8 to 9·9. They are tougher than diamonds but have a certain brittleness. The alloy is cemented on the steel crown in the form of tips, which work requires great care and experience. The bit can be either integral with the rod or detachable, in which latter case it is connected to the rod by a screw thread or wedged in a tapered socket. In some types the bit is inserted in a socket and held in position with a split-pin.

The detachable alloy bit is successfully replacing the rigid carbide bit, since it has a longer life and the drill rods do not require to be of the same high-grade steel which has to be used when the bit is integral with the rod. In the latter case, the bit may have a longer life than the rod itself unless the rods are made from special steel, such as chrome molybdenum steel. With the detachable bit, only the bit requires to be transported for regrinding. With the rigid carbide-

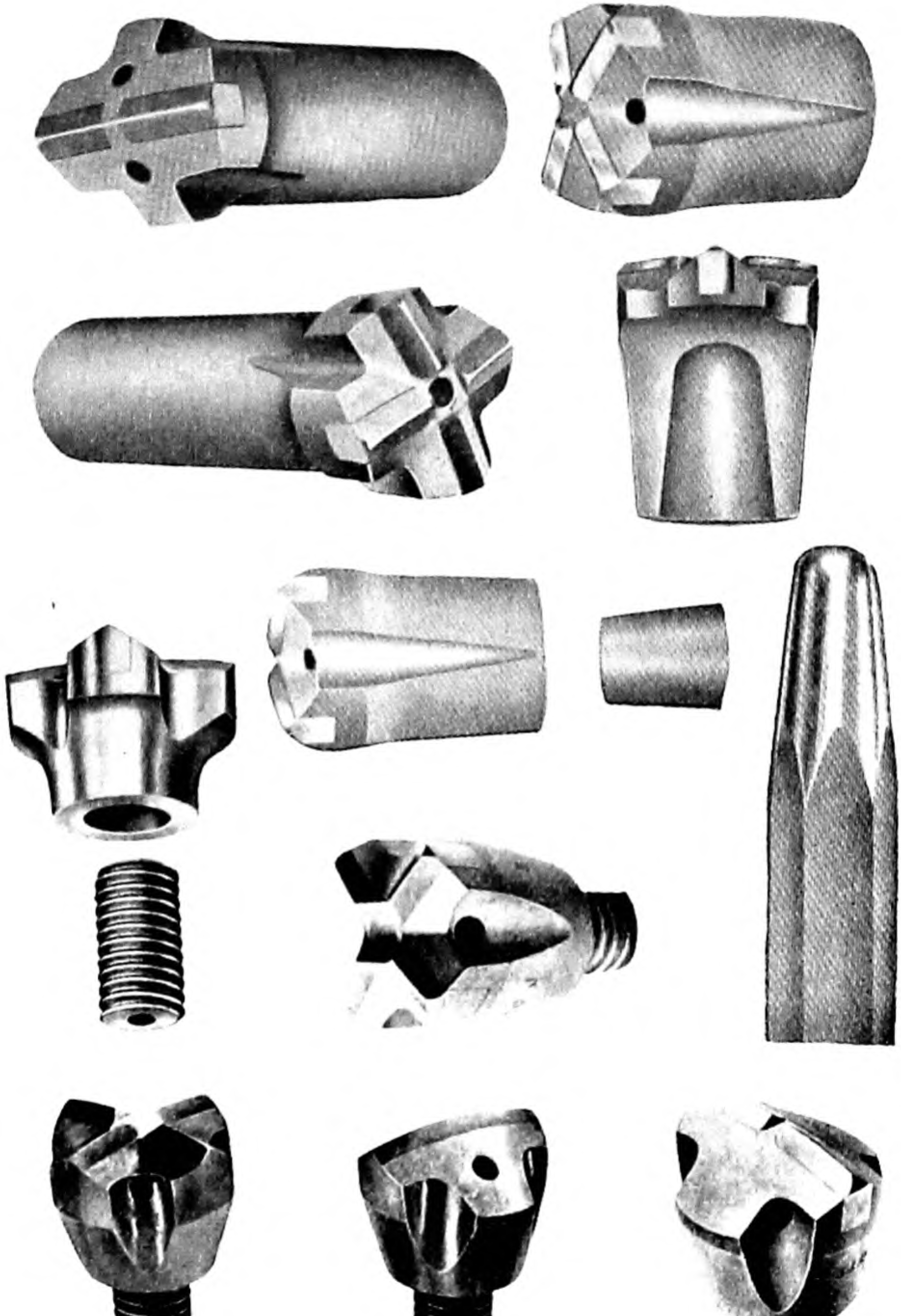


Fig. 126b

tipped bit, the drill rod may fail at the point where vibration stresses cause early fatigue, since the same length of rod is always in use. The design of the cutting edges is usually the simple cutting and cross-bit pattern. Three cutting edges may be used, or even five where the hard ground requires a greater number of cutting edges. By designing the profile of the carbide cutting edges and the body of the bit, the edges can be made to give a fairly long life with regrinding. Similarly, the insertion of the alloy cutting edges in the body of the bit should be carried out in such a way that sufficient space is allowed for the bit profile to be cut away to let the boring chippings get away from the working face. Wet drilling is imperative when using percussive alloy-tipped bits. These bits have many advantages, the first of which is the great increase in the penetration speed. This amounts to from 30–60 per cent. above the speed obtained by ordinary steel bits, and the greatest benefit is found when drilling in the harder rocks, when it may increase to 100 per cent. The alloy bits suffer less loss in diameter and gauge due to lateral wear, and a hole can be drilled with a two-rod set. The regrinding required for the cutting edges is much less than the resharpening of ordinary steel or hardened steel bits.

The main disadvantage of the alloy bits is their high cost, which varies between £4 and £5 each. The initial cost should be set against bit-life, the alloy bit-life being from 10 to 15 times greater than that of a steel bit. The drilling cost using the alloy bits, including the cost of regrinding, will vary between 0·5*d.* and 2·5*d.* per foot according to the type of coal-measure strata being drilled. The bits should be used with a special profile gauge, which will indicate when a regrind is necessary and keep a check on the cutting-edge profile. It is wasteful to drill farther than is indicated, since more alloy insert has to be ground away to give the bit the proper 'set'.

In the Ruhr, a single bit will drill, including all regrinds, 180 to 360 feet in sandstone, 600 to 1,000 feet in sandy shales and 2,000 feet in shale.

(*m*) *Graduation of drilling bits.* The wear and tear of the drilling bits not only takes place on the cutting edges, but also at the sides, and is caused mainly by the rotational movement. The reduction in diameter finally reaches such a degree that the cutting edges cease to be effective. This requires the replacement of the bit with one which is not greater in diameter than the end diameter of the hole,

otherwise the new bit would not fit the hole and would be worn down quickly due to the reaming out required.

During drilling, therefore, a set of drilling bits which are graduated in diameter from one to the other is required. The degree of graduation depends upon the nature of the strata and the probable wear on the bits. In the case of hard strata, a large range may be required, whereas in softer ground, drill steels with a greater graduation in length and bits with a smaller graduation in diameter are used.

The graduation in diameter must be arranged so that the following bit can be put immediately up to the end of the hole without jamming. Experience has proved that with medium hard strata a bit can be used as long as the cutting edge has decreased by not more than 0·08 inch. Allowing a certain tolerance in bit diameter, a graduation by $\frac{1}{8}$ inch is chosen in most cases. With regard to the graduation of drill-steel lengths, it is important that the depth after which the bit must be replaced because of its wear and tear decreases with increasing bit diameter. With a set of drills for deep holes in hard strata it is better to base the determination of the most appropriate lengths for the individual rods on the length of rod for the medium diameter of the hole. Thus, the first drill rods will get a smaller graduation and the final rods a greater graduation of length. If the set requires only a few rods, a uniform graduation is preferred, the average difference in length being 1 foot 6 inches. In the case of harder rocks this graduation decreases to 6–12 inches, while in softer strata the difference in length may be up to 3 feet. It would be possible in the same case to reduce the bit diameter graduation to $\frac{1}{20}$–$\frac{1}{10}$ inch.

As an example of uniform graduation for holes 7 feet 6 inches deep in medium hard strata, the final bit diameter was taken at 1 inch. With a drill rod graduation of 1 foot 6 inches and a bit graduation of $\frac{1}{8}$ inch, five drills of the following lengths were required:

	Length	*Bit diameter*
1st drill . .	1 foot 6 inches	$1\frac{11}{16}$ inches
2nd drill . .	3 feet	$1\frac{9}{16}$ inches
3rd drill . .	4 feet 6 inches	$1\frac{7}{16}$ inches
4th drill . .	6 feet	$1\frac{5}{16}$ inches
5th drill . .	7 feet 6 inches	$1\frac{3}{16}$ inches

Since the penetration speed is inversely proportional to the square of the hole diameter, and all other factors being equal, the same quan-

tity of rock can be removed in the same time, even when drilling holes of different diameter. Holes about 10 inches diameter and more have recently been tried in Germany, but there are still some difficulties with the strength and life of the drilling rods.

(*n*) *Comparison between carbide bits and hardened-steel bits.* Because of the greater hardness of the alloy insert in the carbide bit, the rate of advance which can be obtained in hard and intermediate strata is higher than with hardened steel. The time required to drill a hole is therefore less, including rod changes. This superiority is due not only to the fact that the bit proves to be more satisfactory in the actual drilling performance, but also to the large reduction in lateral wear of the cutting edges and the reduced frequency of drill changing. The reduction in lateral wear of the sides of the bit allows a hole to be started with a smaller initial bit diameter, so that the amount of rock removed by the drill and correspondingly the drilling time, is less. The total operational time per hole is decreased by the reduction in drill changes. The alloy bits may also reduce the number of operators required since, in a drilling rig, two drills may be operated by one man. The necessity for rod transportations outbye is obviated.

The penetration rate with alloy bits is much superior to steel bits in hard strata, but this relative superiority decreases with decrease in rock hardness, so that in soft strata the overall advantages may be reduced in favour of hardened-steel bits. The steel bit in soft strata gains its superiority over the alloy bit, since the cutting edges are more acute and are less likely to entrench in the material and jam, as in the case of the alloy bit. The discharge holes in the alloy bit may be blocked by the softer material. Referring to Fig. 127, the drilling cost with alloy bits is compared with that of hardened-steel bits having equal penetration rates. The cost for the alloy bits still exceeds that for hardened-steel bits due to the higher price of the alloy bit. The greater the difference between the penetration rates obtained under similar rock conditions, the greater is the reduction in this difference until the costs equalise, and then the alloy bit cost is less than for the hardened-steel bit with increasing difference in the penetration rates. The high price of the alloy bit has to be set against the saving in drilling time and the reduction in the total time for the drilling cycle. This must have the effect of reducing the work required and wages cost per foot drilled, and with a high wages

rate the balance will be reached quicker, making the introduction of the alloy bit more economical.

Saving of drilling time also implies a possible decrease in the total time required to complete the drift drivage, assuming that the necessary measures for efficient organisation have also been taken.

Any reduction in the total time schedule will result in a reduction in the cost of the drift, since amortisation and interest on the capital expenditure is reduced. Additional costs, such as for special ventilation and supervision, may also be reduced.

The rate of utilisation of mechanical loading equipment is

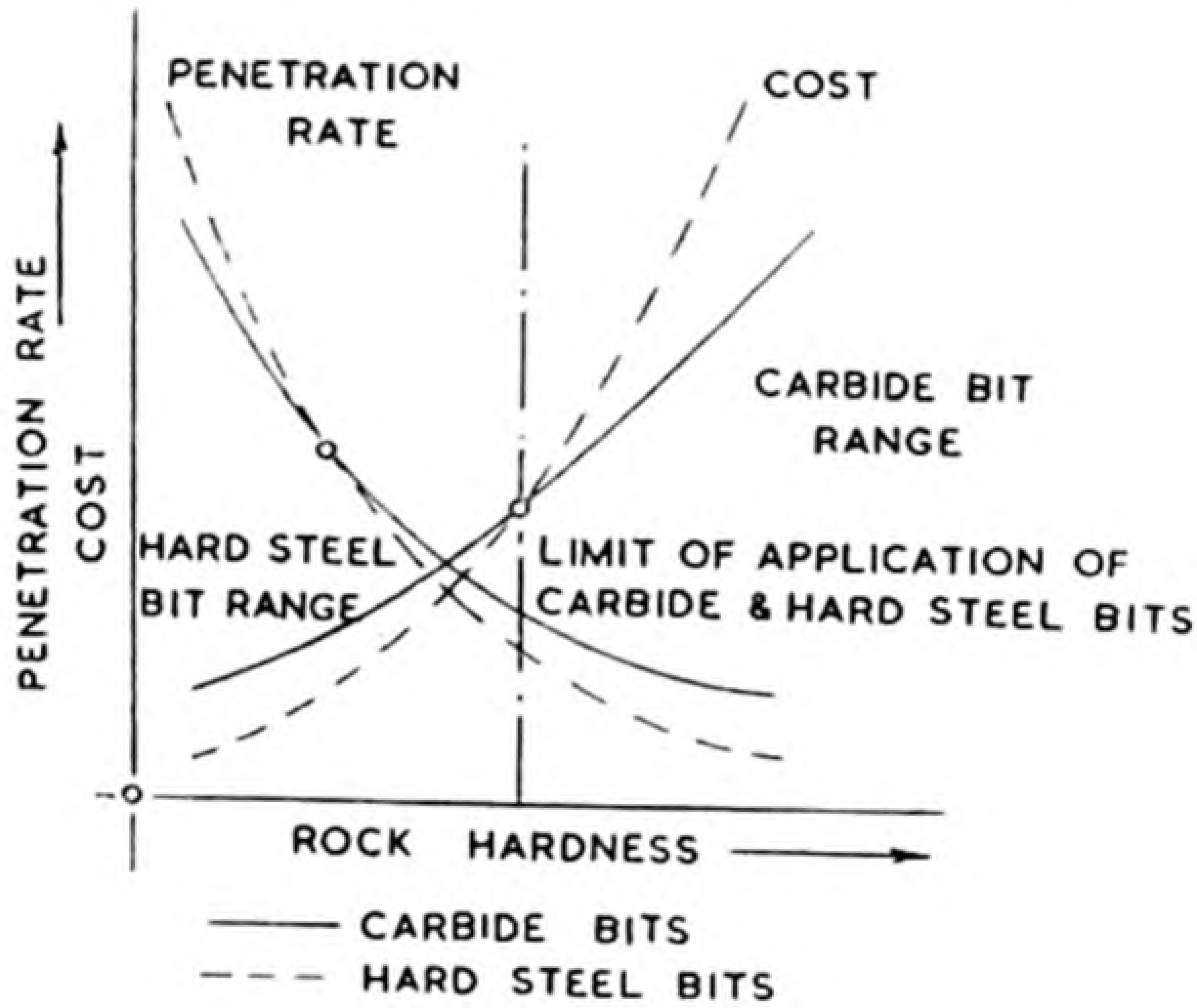


FIG. 127

increased and this cost reduced. In coal-mining the rate of advance due to the introduction of alloy bits has shown an increase of as much as from 8 to 15 per cent., mainly because of the increased drilling speed which can be obtained. The reduction in development time required reduces the development cost by from 4 to 7 per cent. This saving is another factor in favour of the introduction of carbide-tipped bits compared to hardened-steel bits and of a wider extension of their use. The following table * gives details relating to the performance of carbide percussive bits in various rock strata with machine details and a comparison of bit life with hardened-steel bits. The rock hardness has been measured in 'shore' values, using the scleroscope.

* *King's College Mining Bulletin*, Data Sheet No. 3.

Performance of carbide percussive bits in various rock strata

Rock	Location	Nature of Rock	Shore Hardness	Machine	Stroke in. ins.	Bore in ins.	Dead Weight in lbs.	Ft./lb. Blow	Air Pressure in lb. per sq. in.	Bit Grade	Bit Diameter in ins.	Penetration Speed in ins./min.	Footage per Regrind	Footage with Hardened-Steel Bit
Sandstone	Yorkshire Colliery	Abrasive	52	A	1⅝	2¾	31	25	90	I	1·750	5½	405	20
				A	1⅝	2¾	31	25	90	I	1·725	5½	325	20
				A	1⅝	2¾	31	25	90	I	1·700	5½	245	20
				A	1⅝	2¾	31	24	70	I	1·750	8	298	16
				A	1⅝	2¾	31	23½	65	A	1·735	6	92	5
				A	1⅝	2¾	31	23½	65	A	1·700	6	94	5
				A	1⅝	2¾	31	23	60	A	1·737	13	258	14
				A	1⅝	2¾	31	23	60	I	1·470	11	230	14
Red Sandstone	Scottish Colliery	Hard Abrasive	47	B	2	2⅝	48	34	75	A	1·750	4	340	0·5
Granite	Leicester	70% Silica Very Hard	74	B	2	2⅝	48	34	75	I	1·750	6	55	1
				B	2	2⅝	48	34	75	I	2·250	6	70	1
				B	2	2⅝	48	34	75	H	1·750	7	302	1
				B	2	2⅝	48	34	75	H	1·500	7	310	1
Silica	South Wales Quarry	98% Pure Very Hard	97	C	1⅓	2⅝	28	23	70	I	1·500	1	5	0·16
Quartzite	North Wales Quarry	80% Silica Very Hard	85–90	D	1⅝	2⅝	38	20	80	I	1·750	5	40	0·5
				E	1⅝	2⅝	39	25	80	H	1·750	3	11	0·5
				E	1⅝	2⅝	39	25	80	I	1·750	3	31	0·5
				E	1⅝	2⅝	39	25	80	H	1·750	3	43	0·5
Granite Quartzite	North Wales Quarry	84% Silica Very Hard	88–94	D	1¾	2⅝	38	20	80	I	1·750	3½	24	0·5
				F	1½	2⅝	29	22	80	I	1·750	3½	11	0·5
Haematite Iron Ore	Cumberland Mine	Hard Kidney Ore	72	G	2½	2⅝	51	30	75	H	1·500	10	119	0·16
				G	2½	2⅝	51	30	75	H	1·500	10	328	0·16

H—carbide designed for use in hard rocks. 6–7½% Cobalt.

I—carbide designed for use in intermediate rocks. 7½–9% Cobalt.

A—carbide designed for use in abrasive rocks. 9–11% Cobalt.

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Nature of Rock	*Locality*	*Mohr Hardness*	*Scleroscope Hardness*
Red Flint	Wales	8	99
Chert	Wales	8	97
Greenstone	Penventon	7	92
Quartzite	Clunie Tunnel, Scotland	8	88
Black Rock	Kolar Goldfield, India	8	84
Granite (Blue Grey Elvan)	Penleigh Quarry, Cornwall	7	82
Granite	Troon Quarry, Cornwall	7	81
Granite (Elvan)	St. Keverne, Cornwall	7	79
Granite	Sweden	7	77
Granite (Greenstone)	Cornwall	7	76
Granite	Geevor, Cornwall	7	76
Granite	Guernsey Quarry	7	76
Granite	Leicester	7	74
Iron Ore (Haematite)	Cumberland	6	72
Granite	Norway	7	71
Granite (Grey Elvan)	Carn Brea, Cornwall	7	71
Mica Schist	Clunie Tunnel	6	69
Sandstone (Blue Pennant)	Derbyshire	6	63
Sandstone (Parkgate)	Barnsley	5	58
Sandstone (Hazel)	Stanhope, Co. Durham	5	53
Limestone	Derbyshire	5	53
Sandstone (Parkgate)	Dinnington, Yorks	5	52
Schist	Clunie Tunnel	6	48
Sandstone	Carmarthen, Wales	5	46
Sandstone (Hutton Post)	Co. Durham	5	43
Portland Stone	Gypsum Mine, N. Cornwall	2	32
Shale Band	Hutton Seam, Durham	2	23

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(*o*) *Rotary drilling machines.* Rotary drilling is the most prevailing method for drilling coal, salt and soft iron ore. It has been in use for soft and medium-hard stone for many years. Soft and medium-hard stone can be bored with a hand-operated rotary drill, the drive of which can be electrical or by compressed air. These machines are suitable for all cases where a drilling pressure of about forty pounds is sufficient to drive the cutting edge into the stone.

An electrical hand-operated drilling machine consists principally of a three-phase squirrel-cage motor housed in a light metal casing which incorporates two large curved handgrips, one of which contains a spring-loaded lever operating a switch, affording positive hand control of the machine. The machine weighs between 35 and 40 lb., according to its make and speed. In the front of the motor

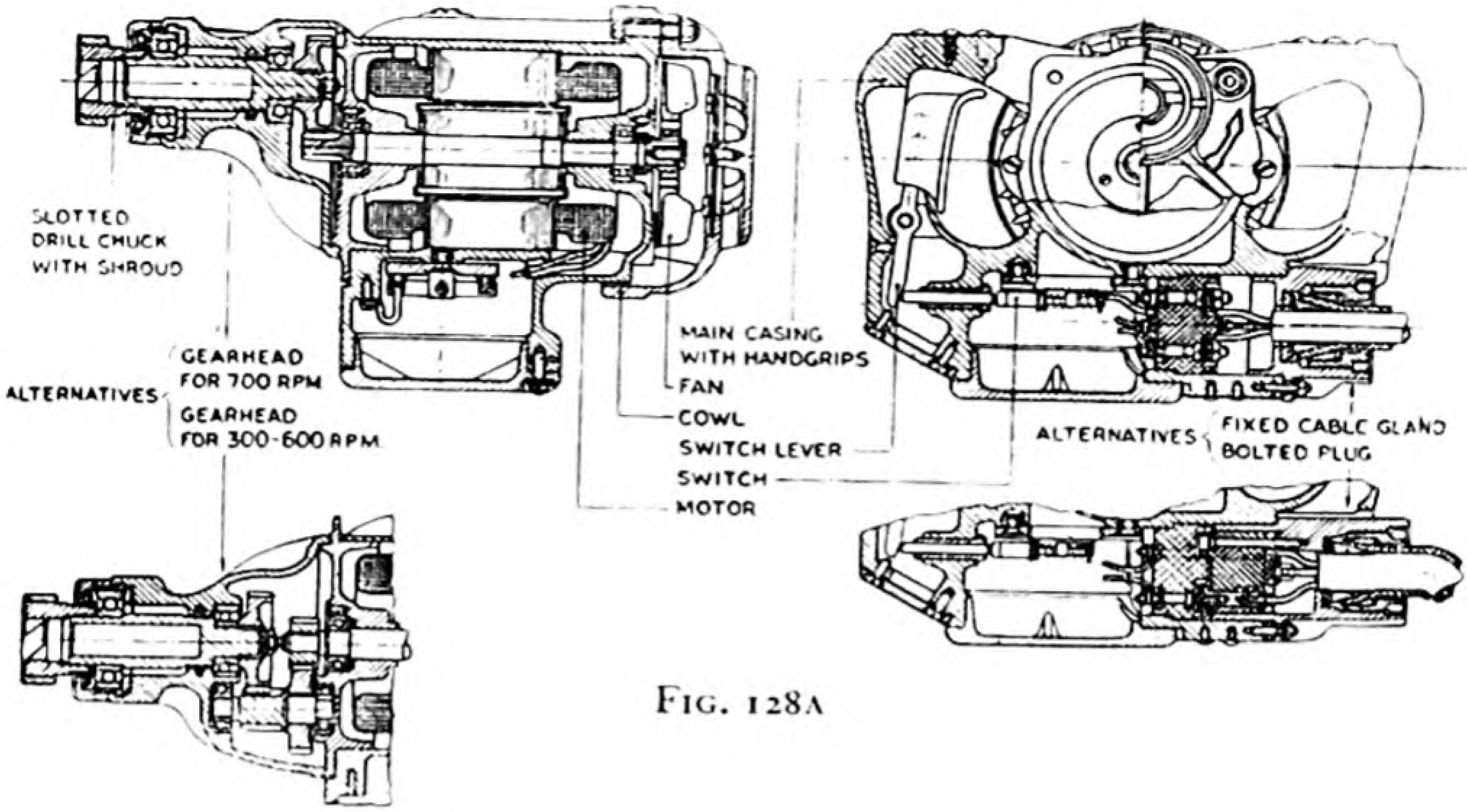


Fig. 128A

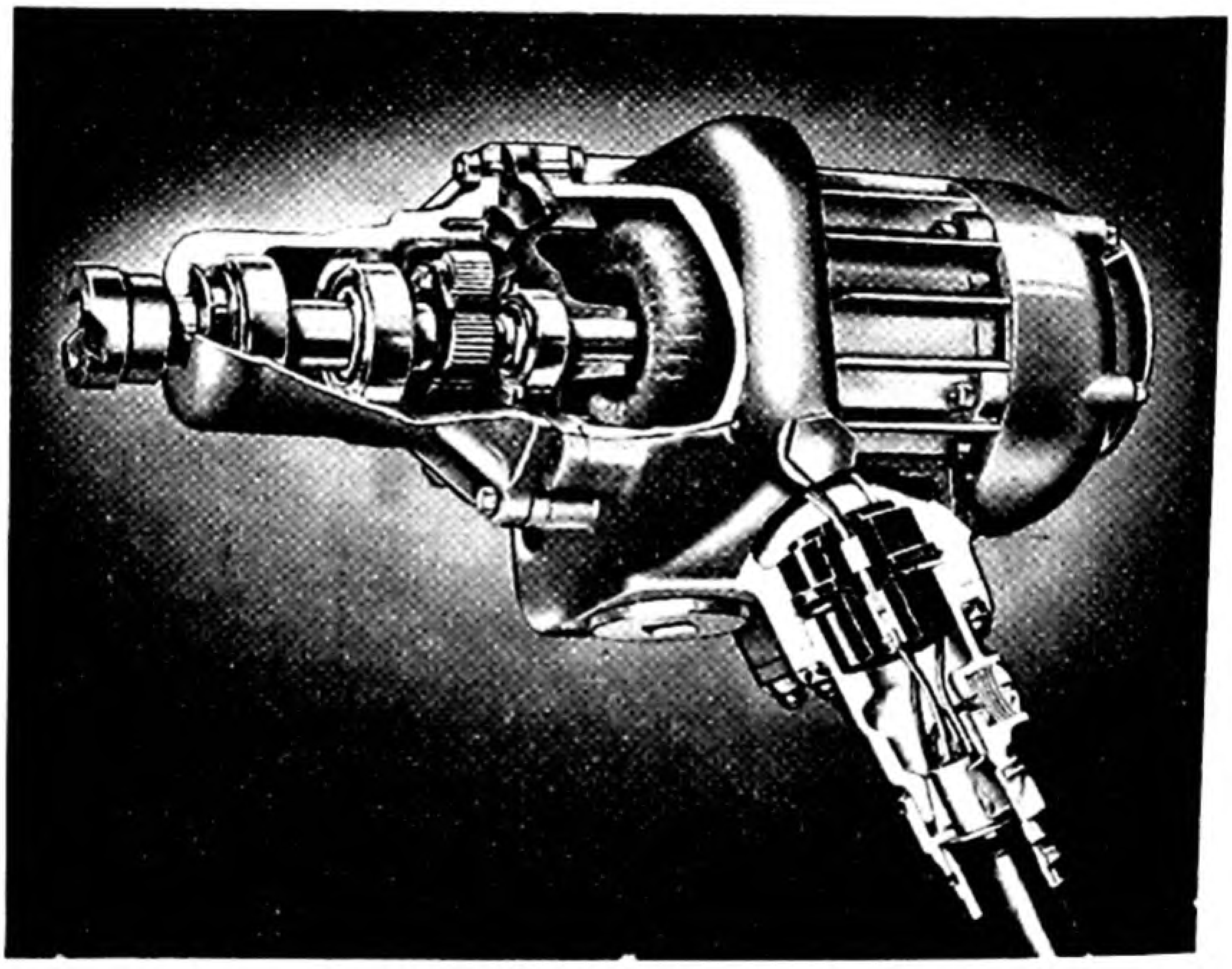


Fig. 128B

casing is the gear box, which accommodates a slotted chuck to take the drill rod. The switch is usually of the single-pole type for remote control, while a cable outlet is provided with means for holding the cable firmly. The main features of such a machine are shown in Figs. 128A and B. The machine is usually provided with external fan cooling, which improves the performance of the drill very considerably

by allowing the motor rating to be increased and rendering the operation of the machine more comfortable for the driller. The motors used vary from 1 h.p. to 1½ h.p. and are limited by regulation to operate at a maximum of 125 volts. Some drills are designed to operate on 150 cycles and others on the standard 50 cycles supply.

FIG. 129

The high-frequency machine requires a rotary frequency change in the drill panel and, being a rotary machine, necessitates more maintenance than the normal static transformer. The gearing adopted will give drill speeds of from 600 to 700 r.p.m. for coal, or lower speeds of between 300 and 600 r.p.m. for stone. Various gear heads, as illustrated in Fig. 129, are usually provided to suit local requirements. The third gear head illustrated is a low-speed gear giving 180 r.p.m. for use with mechanical feed equipment. Many collieries prefer to

standardise on a 'general purpose' machine with a speed range between 400 and 600 r.p.m. for both coal and stone.

The following tables illustrate the performance of one type of rotary electric drill in coal and stone when drilling $1\frac{11}{16}$-inch holes *:

Subject drilled: Coal Height of coal: 4 ft. Speed of drill: 430 r.p.m. Operation: hand-held (one man)	Number of holes: 80 Depth of holes: 6 ft. Diameter of holes: $1\frac{11}{16}$ in. Number of rods: 2 (3 ft., 6 ft.) Number of bits: 2
Summary	$\dfrac{\text{Actual drilling time}}{\text{overall time}} = \dfrac{\text{2 hrs. 1 min.}}{\text{3 hrs. 42 min.}} = 54\%$ Av. drilling time per 6-ft. hole = 90 sec. Av. drilling rate = 4 ft. per min.

Subject drilled: Stone: medium hard Speed of drill: 430 r.p.m. Operation: hand-held (two men)	Number of holes: 8 Depth of holes: 6 ft. Diameter of holes: $1\frac{11}{16}$ in. Number of rods: 2 (3 ft., 6 ft.) Number of bits: 4
Summary	$\dfrac{\text{Actual drilling time}}{\text{overall time}} = \dfrac{\text{22 min. 13 sec.}}{\text{39 min.}} = 57\%$ Av. drilling time per 6-ft. hole = 2 min. 46 sec. Av. drilling rate = 2 ft. 2 in. per min.

Subject drilled: Medium grey sandstone over 5 ft. coal seam Speed of drill: 180 r.p.m. Operation: mechanical, with rack feed	Number of holes: 8 Depth of holes: 4 ft. 6 in. to 5 ft. Diameter of holes: $1\frac{11}{16}$ in. Number of rods: 3 (2 ft. 3 in.) (4 ft. 6 in.) (5 ft. 6 in.) Number of bits: 4 Stand: adjustable 6–9 ft.
Summary	$\dfrac{\text{Actual drilling time}}{\text{overall time}} = \dfrac{\text{42 min. 15 sec.}}{\text{135 min. 15 sec.}} = 30{\cdot}6\%$ Av. drilling rate = 10·8 in. per min.

With mechanical feed, considerably harder stone can be drilled than with the hand-held machine. Several mountings with mechanical feed are shown in Figs. 130 and 131. Certain hard rocks cannot, however, be drilled with these small machines even with the drifter

* Rotary Electric Mining Drills, Flax, *K.C.M.S. Journal*, xix.

mounting illustrated, since the bit wear would be so great as to render it impracticable. In such cases special rotary rock drills with force feed must be used.

The drill rods used are of two types, known as 'turbine' and

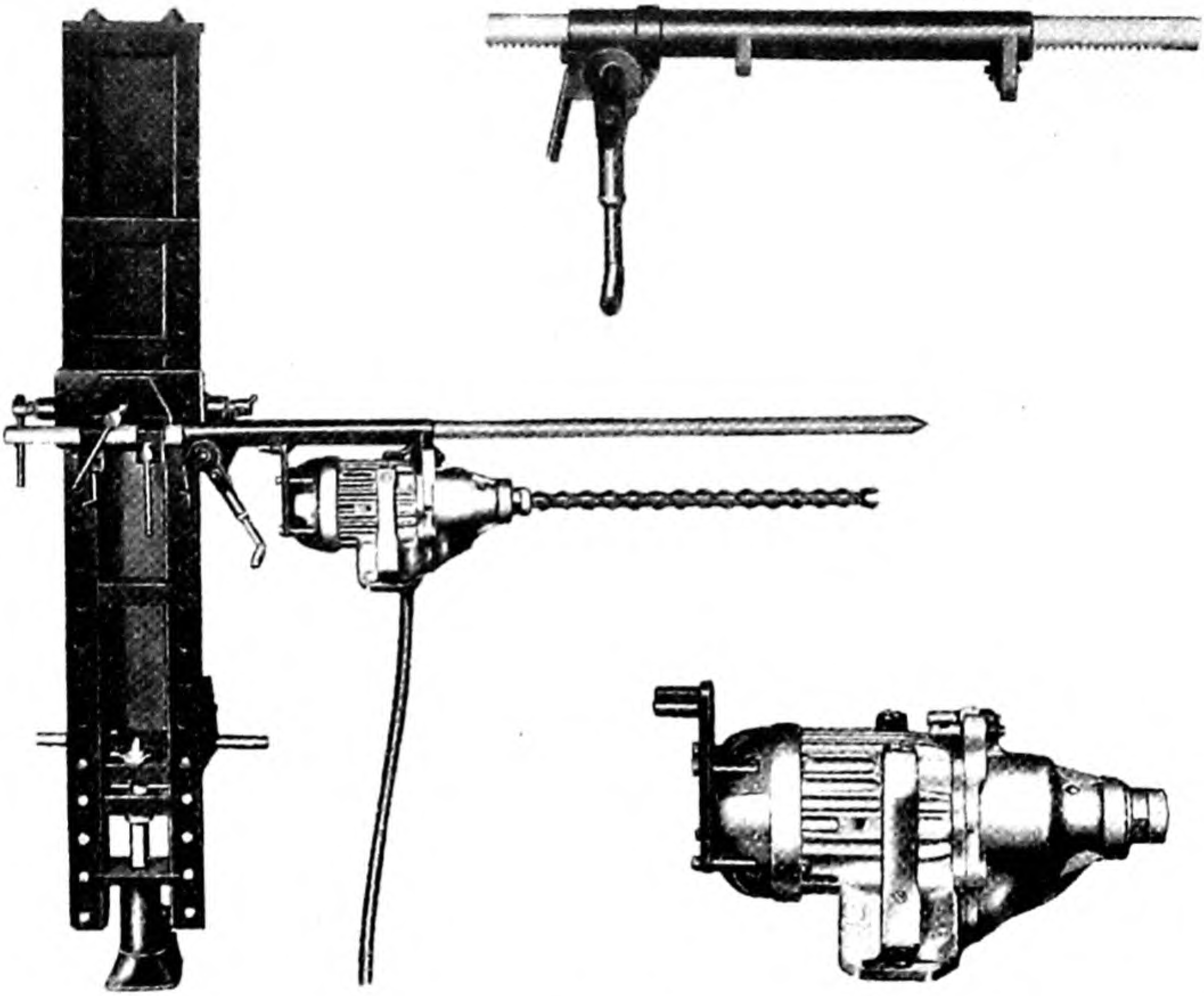


FIG. 130

'diamond' sections, these being self-explanatory. The turbine section is generally regarded as the stronger and wears better than the diamond type, as it is thicker in the scroll. Where wet drilling is applied, a water coupling can be adopted, the chuck being provided with a branch connected to a water-supply, as shown in Figs. 132 and 133. The driller controls the water flow through a stop-cock. The rapid developments made in the design and manufacture of the drilling bits has determined the extent to which the small rotary

drill is used. The introduction of the carbide-tipped bit has greatly increased the effective capacity of rotary drilling. The general form of the bit used is a two-winged steel body with a tungsten carbide

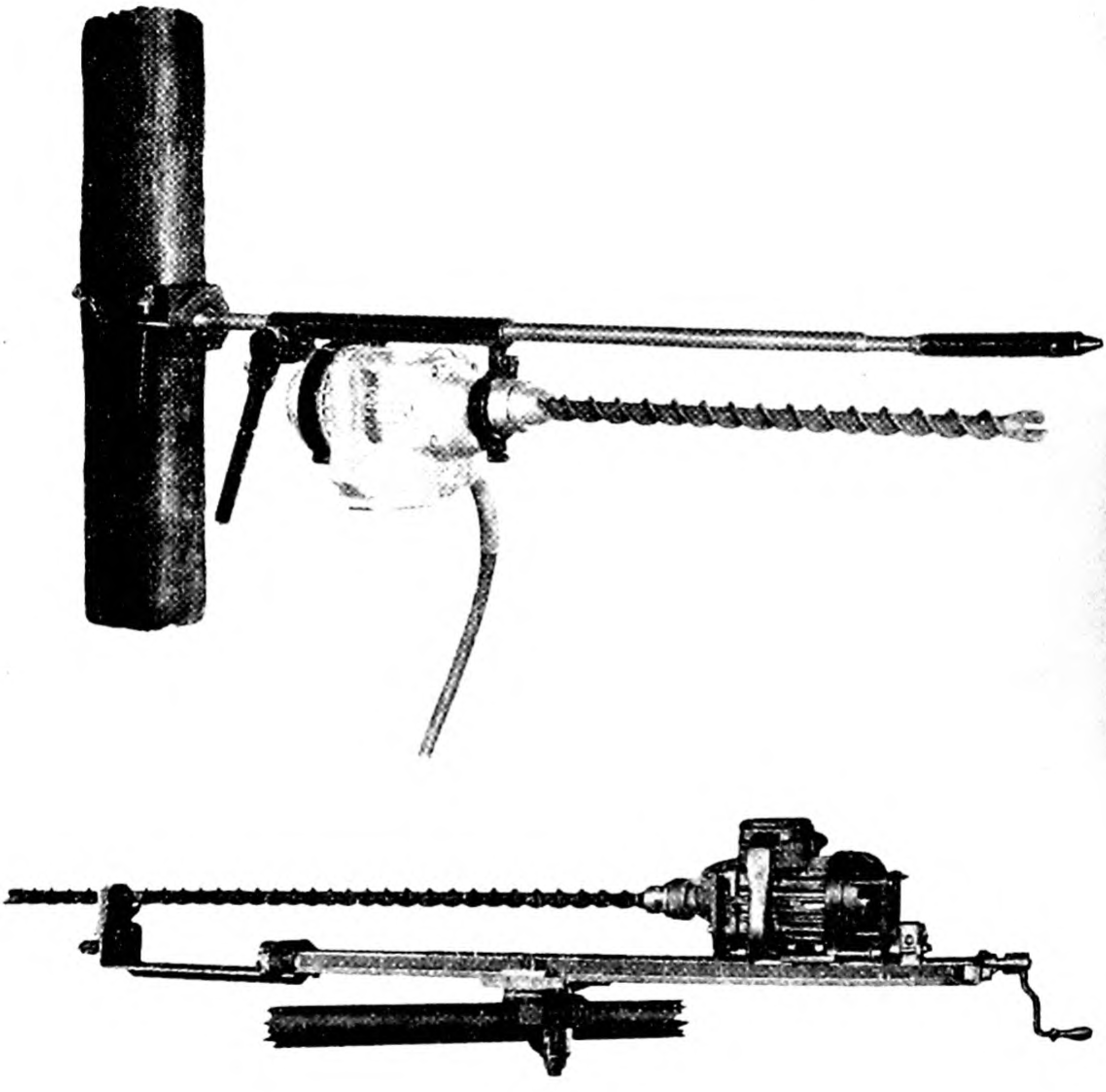


FIG. 131

plate, or tip, secured to each wing. This bit is in two shapes: in the eccentric type the tips are at unequal distances from the centre line of the bit so that the tips cut in different circles, and in the concentric type the tips are set at equal distances from the bit centre line. In the

former, the core is left to break off as drilling proceeds, while in the latter, the core is of 'V' shape and is cut away as the drilling proceeds.

FIG. 132

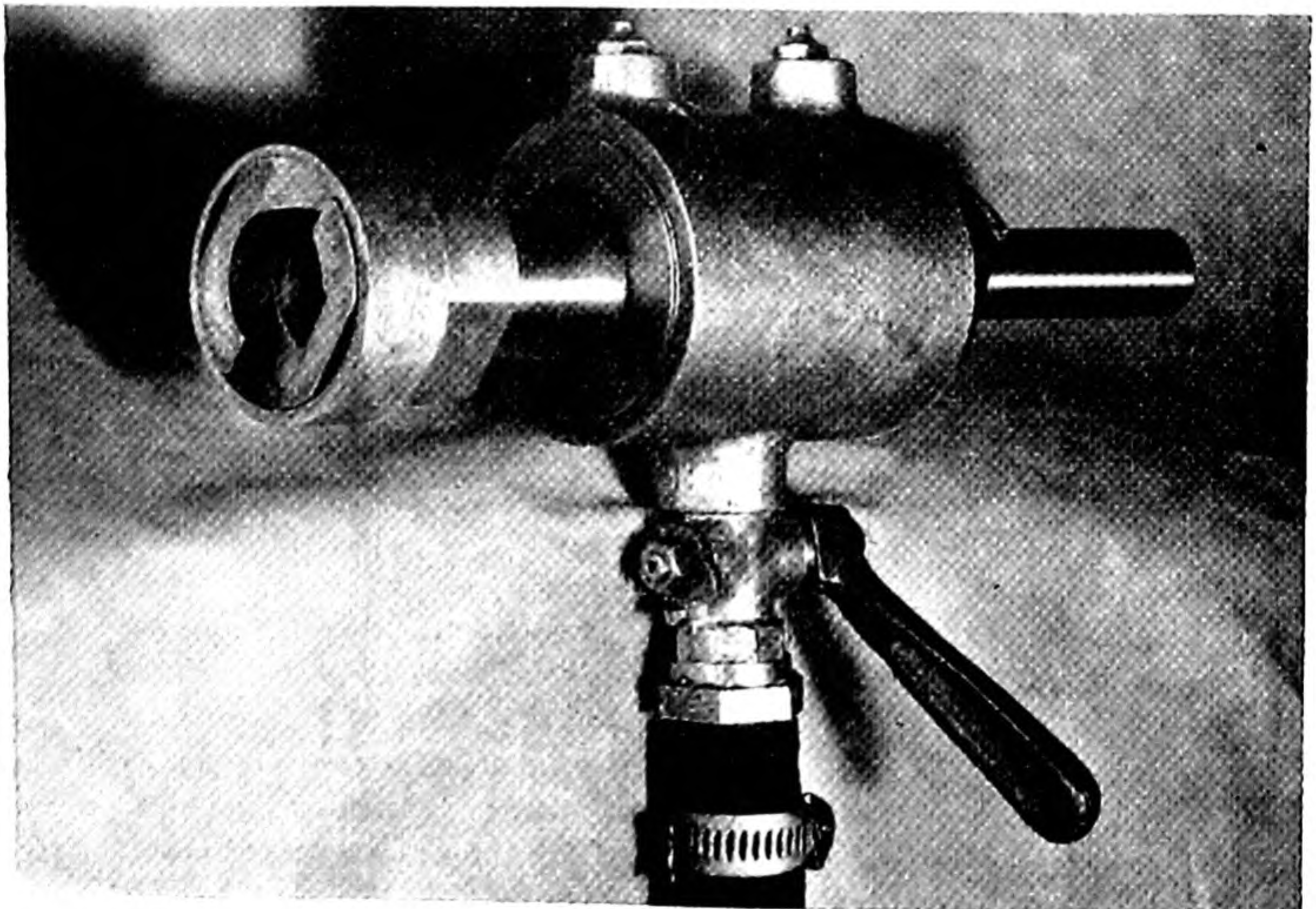


FIG. 133

There have been many modifications in wing profile and grade of insert. There are usually three grades of carbide insert for particular tasks. Standardisation in shape and grade is a factor at collieries

where such drills are in use. The medium grade with eccentric points is predominant. Fig. 134 illustrates a series of carbide-tipped rotary drilling bits. The bit diameter is usually $1\frac{11}{16}$ inches, but larger bits of the same pattern up to $2\frac{3}{4}$ inches diameter are in use. Larger diameter holes up to 4 inches are usually reamed out with a bit in which fixed tips, or detachable single wing bits suitably spaced, are used.

In some mines where the available power is compressed air, compressed-air rotary drills may be used. The use of carbide bits with rotary drilling machines operated by compressed air has made it possible to introduce a light machine, such as illustrated in Fig 135. The machine used for stone- and coal-drilling operates at from 230 to 350 r.p.m. The machine illustrated in Fig. 135 is provided with a double reduction gear, giving a nominal drilling speed of 350 r.p.m. The construction of the rotor holding the plastic vanes is shown in the section in Fig. 135. The vanes are thrown outwards by centrifugal force in the retaining slots in the rotor and keep in contact with the liner. The rotor is set eccentrically to the inner casing of the drill, the space between the rotor and the liner acting as an expansion chamber until each vane reaches the point of exhaust.

The adaptation to wet drilling is carried out by fitting a special wet-drilling attachment and using hollow rods. This attachment is separate from the drill and there is no danger of air coming into contact with the water inlet. The adaptor is fitted with a special chuck to allow for the thickness of the collar, and contains a sealing ring which grips the drill rod and prevents water leakage past the rod and chuck.

Recently, rotary drilling machines have been developed which are also suitable for hard-rock drilling. The decisive factor is a sufficiently high drilling pressure, enabling the cutting edge to penetrate the strata. During the rotating movement, the drilling bit acts as a cutting edge on a lathe and the borings should be in the form of chips. This decreases the risk of silicosis. If the drilling pressure is too low, the cutting edge only grinds the bottom of the hole, resulting in small progress and rapid wear of the bits. The cutting edge needs some time to overcome the inner friction of the stone particles, and therefore the turning speed should be restricted to from 100 to 300 r.p.m., depending upon the hardness of the rock.

TYPE 'S'

A single point tip across the whole width of the bit. Used for exceptionally hard stone with a mechanically fed drill; particularly suitable for starting a hole, to be followed through with the types 'R' or 'KE'.

TYPE 'KE'

The core is parallel but the tip points concentric, avoiding short cutting edges. Frequently more suitable than the type K for abrasive subjects, particularly with mechanically fed drills.

TYPE 'R'

Each tip has a long cutting edge, and the core is small. Used for harder grades of stone with a mechanically fed drill.

TYPE 'K'

The tip points are eccentric and the core parallel. A General Purpose Bit for coal and medium hard stone.

FIG. 134

The necessary high drilling pressure, from 1,800 to 4,500 lb., can be produced only by a special feed motor. This motor and the drilling motor are driven by compressed air, as electrically driven motors have not yet been developed. On account of their heavy weight and the necessary high drilling pressure, these rotary drills can only be used mounted on a drilling rig. Generally, one rig carries two drilling machines. The actual performance in sandstone will be

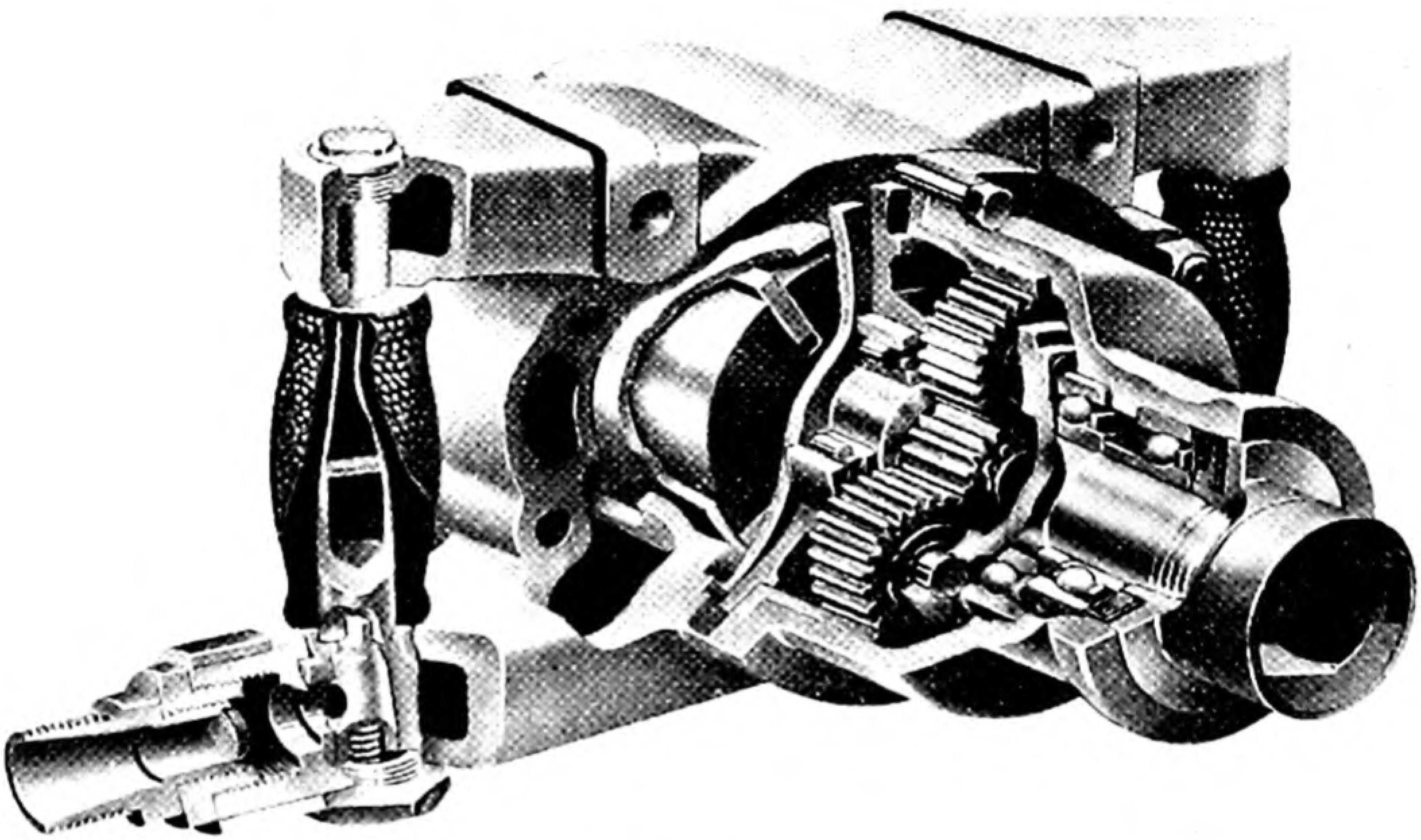


Fig. 135

from 20 to 35 inches, in sandy shale from 30 to 50 inches and in shale from 40 to 80 inches per minute.

Further advantages of rotary drills are less noise, small dust production and lower operational costs.

In very hard rock, rotary drilling has not yet been successful.

Section 2. Rock Loading

(*a*) *Introduction.* The loading operation in drift driving in coal mines is being mechanised in many instances where high speed has been found essential and where there is adequate transportation available for the disposal of debris from the face. There are numerous machines in use, all of different type and method of operation, each having their advantages and disadvantages. There are, how-

ever, a great number of drifts and roadways in which manual loading is still carried out, since in these cases high speed is not essential.

(*b*) *Hand-loading*. In the hand-loading of the debris from a drift face, time studies have indicated that the actual manual loading work involved takes up only 50 per cent. of the available time, while 15 per cent. is required for tub-changing, 13 per cent. for supplying empty tubs to the face and transporting full tubs from the face, 9 per cent. for auxiliary work and about 10 per cent. for rest periods. The average loading result per man-hour depends on the nature of the stone and the height of the tub, other conditions such as number of men being equal. Under normal conditions, when loading sandstone, an efficiency of from 1·3 to 1·7 cubic yards of debris per man-hour can be anticipated. This efficiency will increase when loading shales by about 10 per cent. to from 1·4 to 1·8 cubic yards per man-hour. With an overall working time of six hours, an average output of 9 cubic yards per manshift can be achieved in sandstone and 10 cubic yards per manshift in shale. The decrease in hand-loading efficiency will be about 6 per cent. for every 4 inches of tub height above 3 feet and up to 5 feet. The number of men employed on a drift face will influence the loading capacity per manshift adversely if there is hindering of the work due to restriction or realisation of the maximum capacity by full utilisation of the labour involved. Under normal drift-size conditions, the best loading capacity per manshift is achieved with two men. When three men are employed, there is a decrease of about 15 per cent. in the loading capacity per manshift, although lost time on tub-changing may be reduced. It is important to introduce an easy system for tub-changing. If double track is used, it is easy to change tubs, provided the empty tubs are kept well up to the face. Where single track is in use, it is advisable to install a portable track section capable of accommodating from seven to ten tubs and coupled to the main track with a portable switch. Alternatively, a steel sheet can be used to traverse the tubs from the empty to the full track.

(*c*) *Mechanical loading*. The hand-loading of 1 cubic yard of debris takes approximately 35 to 45 working minutes; this means that, per cubic yard of solid rock, for shale 50–65 and for sandstone 65–85 working minutes are needed. The time for all operations in a hand-driven drift may amount to from 150 to 230 minutes per cubic

yard of solid rock, of which the proportion taken up by the loading operation will be between 30 and 40 per cent. Mechanical loading has been introduced to obviate the delays arising in hand-loading methods, reduce the time occupied by the loading operation and reduce the total drivage cost. The loading operation may be partly mechanised by introducing a *gate end loader* on to which the debris is hand-filled and conveyed into tubs or on to a belt conveyor. This partial mechanisation does not materially increase the overall drivage per manshift, and full mechanisation of rock loading is preferred.

The scraper loader (slusher) is a simple and robust machine consisting essentially of the scraper bucket, the scraper engine or hoist with the haulage drums, the runway or slide, and the ropes with the return sheave.

The slide is mounted on a wheeled frame, while the ropes from the haulage drums pass round two pulleys in the rear end of the jib. The main rope is connected to the front end of the scraper bucket, while the tail rope passes round a sheave secured at the face and is fastened to the rear end of the scraper bucket. The main rope pulls the loaded scraper bucket from the face up the slide so that its contents are dropped through the jib into the tub. The tail rope pulls the bucket back to the face for reloading. The machine illustrated in Fig. 136 is an air-operated Sullivan slusher, which has three controls, viz. the valve on the air line and the two operating handles A and B. The operating handles control the friction clutches on the main and tail rope drums. The levers operate independently and in the same direction. The loader is provided with two clamps, one on each side of the machine, designed to grip the head of the rail. The following details show the leading dimensions of the machine, which may be operated by electric motor or compressed air:

Width of jib . . .	3 feet 9 inches.
Overall width of apron .	4 feet 7 inches.
Height of jib . . .	5 feet.
Headroom required . .	6 feet.
Overall length . . .	20 feet 8 inches.
Track gauge . . .	2 feet (variable).
Capacity	30–40 tons per hour.

The scraper bucket employed has normally a capacity of 0·4–2·0

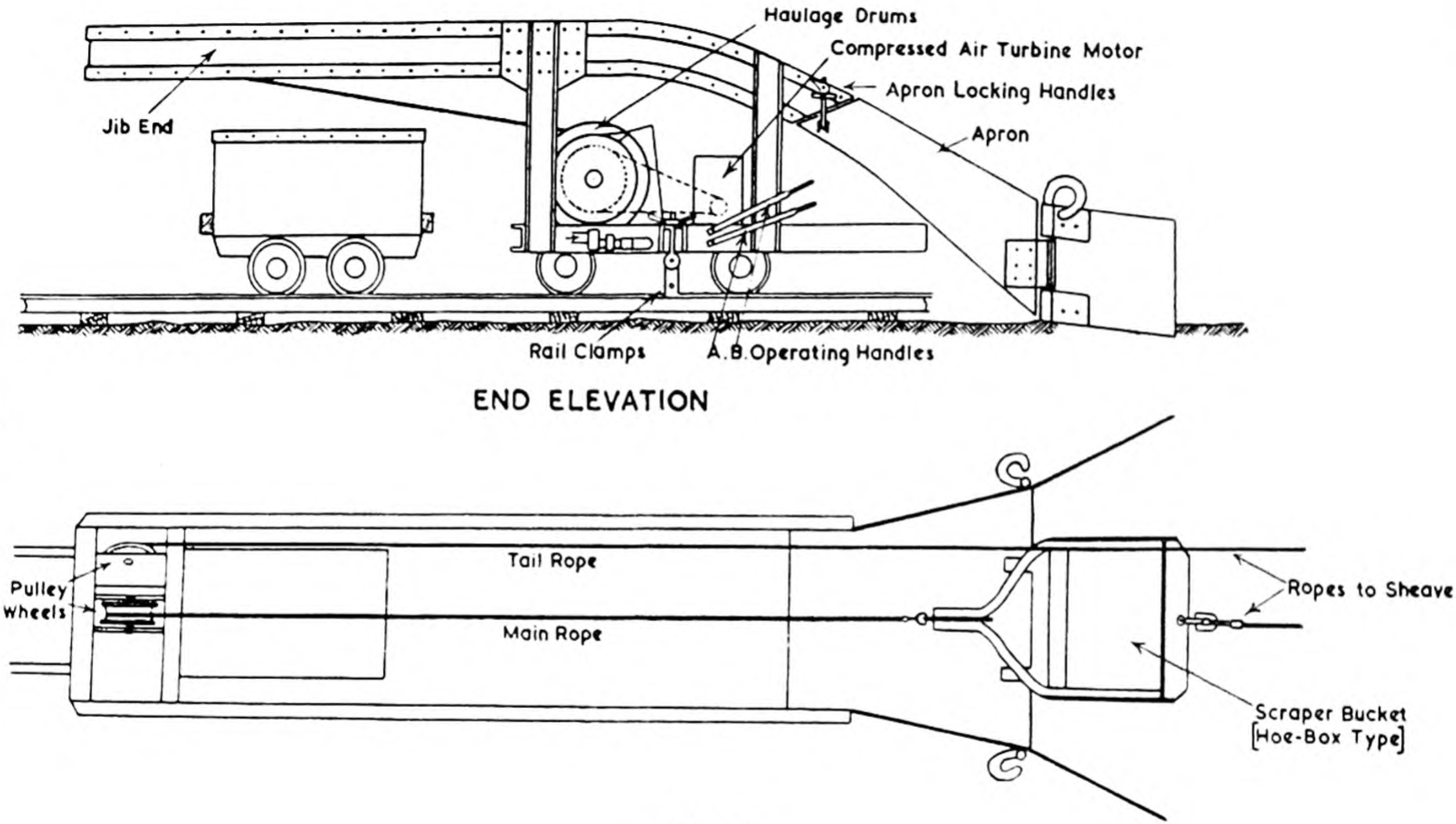
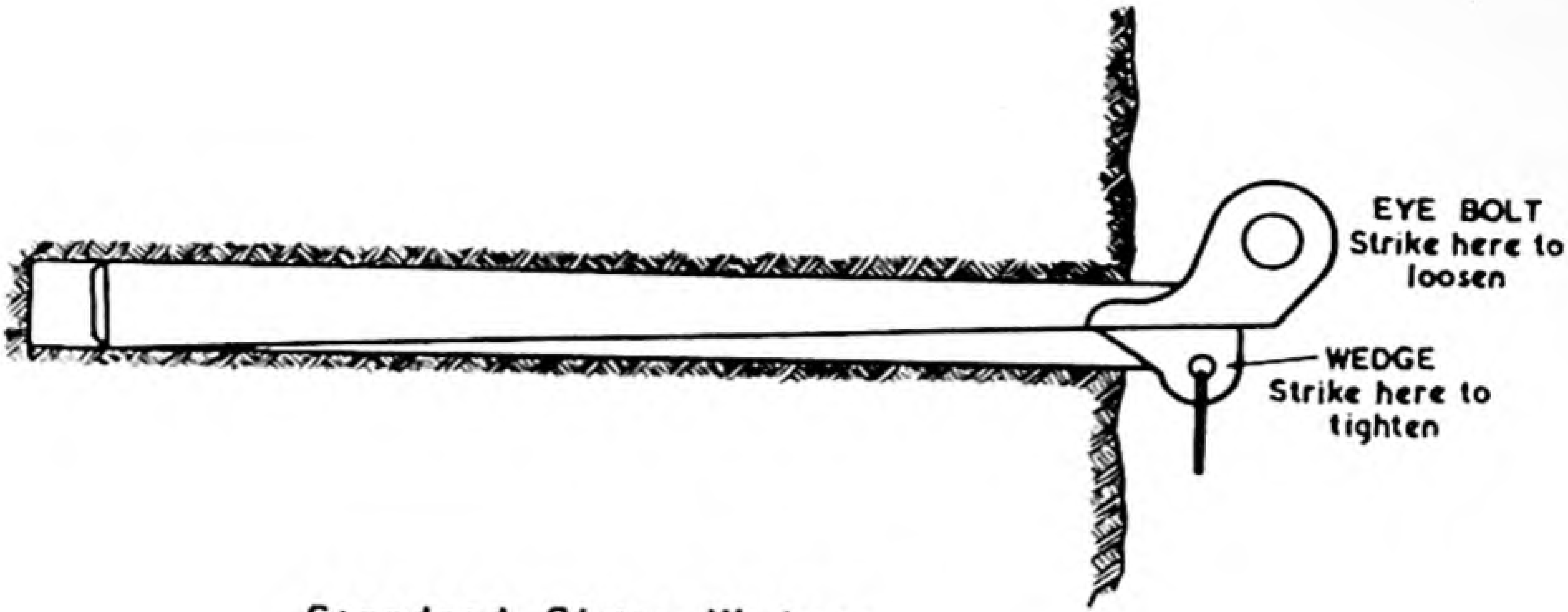
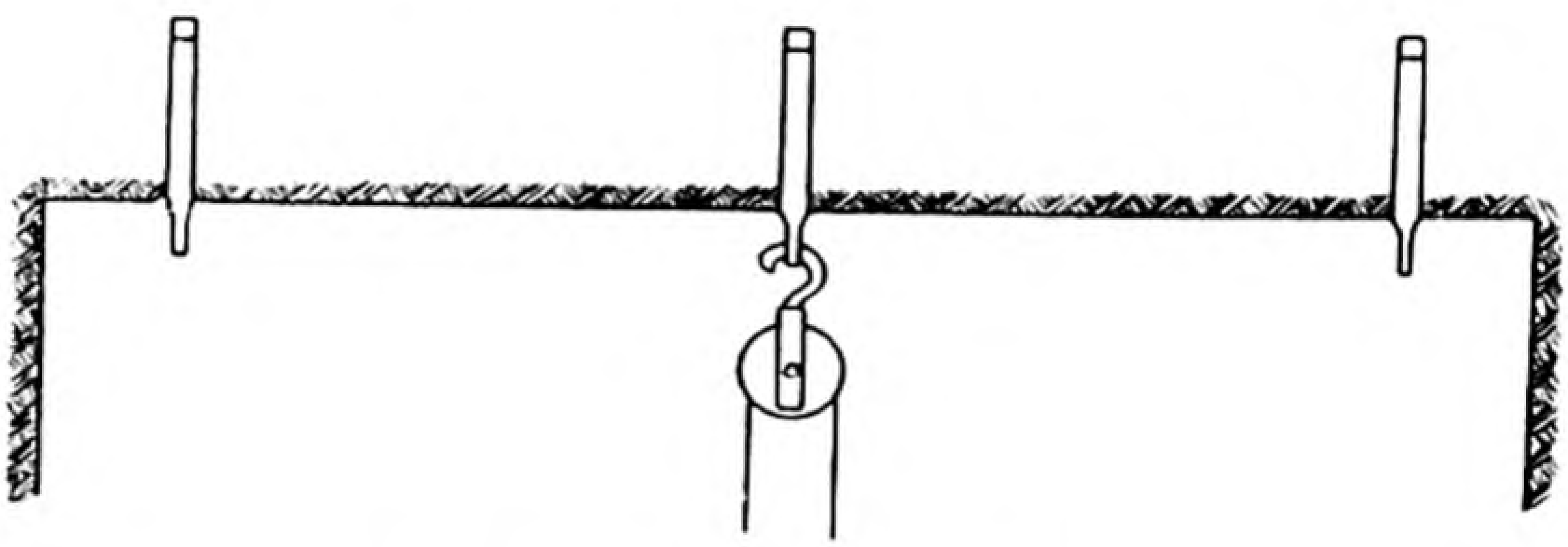

Haulage Drums
Compressed Air Turbine Motor
Apron Locking Handles
Apron
Jib End
Rail Clamps
A.B.Operating Handles
END ELEVATION
Tail Rope
Main Rope
Pulley Wheels
Ropes to Sheave
Scraper Bucket [Hoe-Box Type]

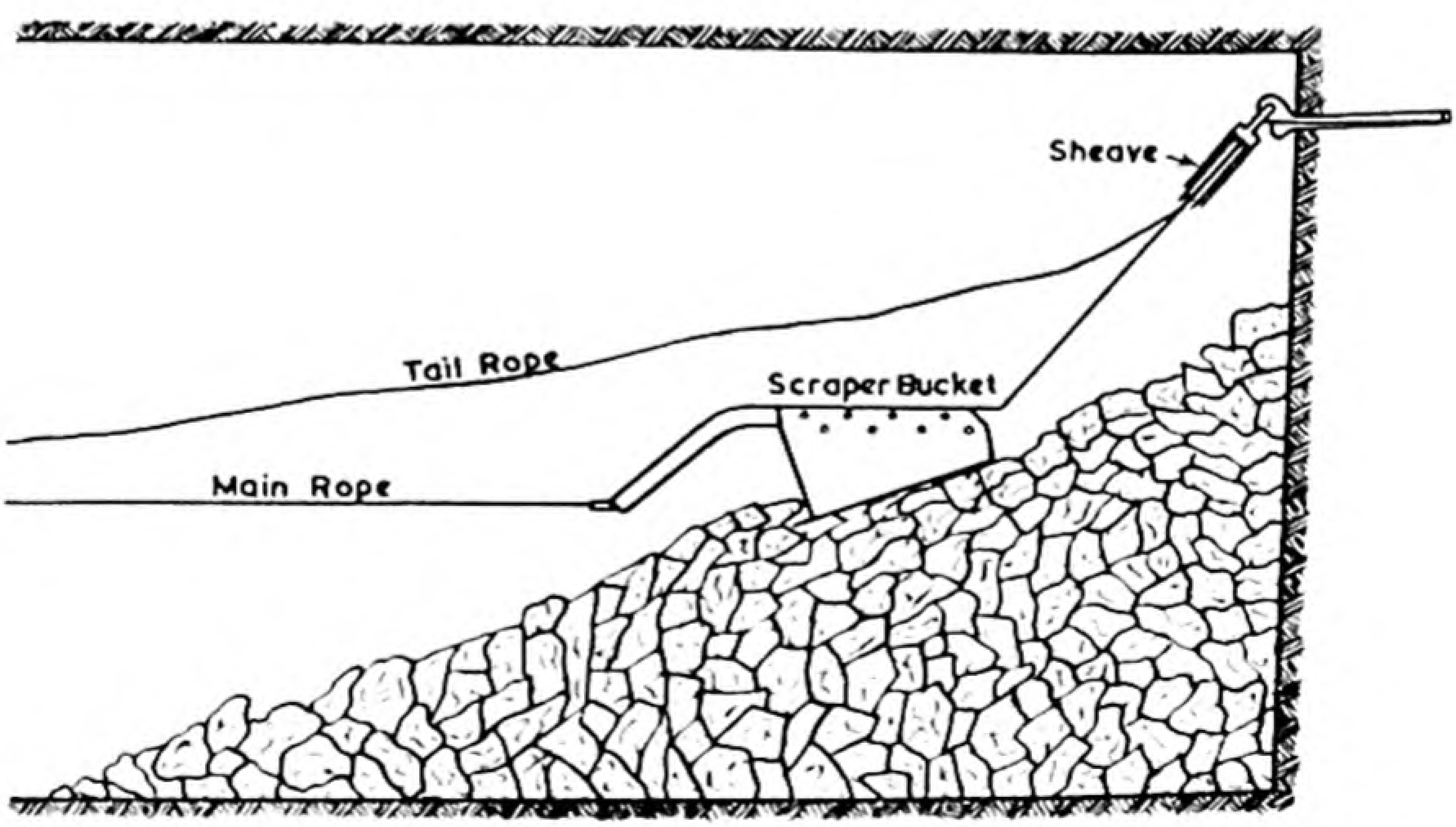
FIG. 136



Standard Sister Wedges



Arrangement of Sister Wedges



Section of Face during Loading

FIG. 137

cubic yards, and is usually of the hoe-box type fitted with three cast-iron counterweights, which may be removed if the digging characteristics of the bucket are required to be altered. The lip plate is of manganese steel, with two sets of fixing holes to enable it to be lowered when it has worn down to the bucket itself.

The return sheave is hung at the face of the headings from sister wedges, as shown in Fig. 137. These wedges are 2 inches diameter and require a hole 20 inches deep, which may need to be longer if the ground is broken. Three holes are required for varying the position of the sheave, which are usually sufficient to cover the whole width of the drift. In operation, the slusher is securely fixed by the rail clamps, the sister wedges fitted and the sheave hooked to the centre position. The operator starts the machine, which pulls the scraper bucket to the top of the pile of muck or stones in the face when the motion is reversed, the bucket digs into the pile and loads out over the ramp on to the jib and into the tub. The process is repeated until a gulley has been made in the rock pile down to floor level and the bucket can no longer pick up a load. At this stage the sheave is transferred to one of the side wedges and, when that side of the drift has been cleared, moved to the other side. To deal with the stones next to the face, the scraper is pulled up to the sheave and then dropped vertically, and in this way it is possible to load approximately 95 per cent. of the stones. The scraper should always be filled before pulling to the slusher, since empty or partially filled scrapers will tend to dig up the floor, while the output is obviously reduced.

At the commencement of loading operations the unit is secured to the rails by the rail clamps at about 7 yards from the face, the sister wedges fitted and the sheave hooked to the centre wedge. The slusher can remain in this position until the face has been advanced to approximately 15 yards, when it is desirable to move the slusher nearer to the face. If the distance from the face to the slusher becomes too great, the increase in scraping distance will affect the capacity of the machine. When shot firing, it is not necessary to move out the slusher, since the haulage is adequately protected by the ramp. The distance from the landing to the slusher should be kept as short as possible in order to keep the tub-changing time to a minimum.

The personnel required to operate the slusher are usually as follows:

1 man operating the loader.
1 man shovelling and 'barring down' at the face.
2 men supplying tubs from the landing or standage.

A fifth man will be required to operate the auxiliary rope haulage engine for pulling the full stone tubs into the landing, where this method of haulage is in use.

Loaders with gathering arms. Several models of this type of loader are available, the most recent British model being the M.C.2, which has been designed specifically to eliminate as far as possible all sources of breakdown, and is an improved version of the original M.C.8 BU.4. The loader may be fitted for either electric or com-

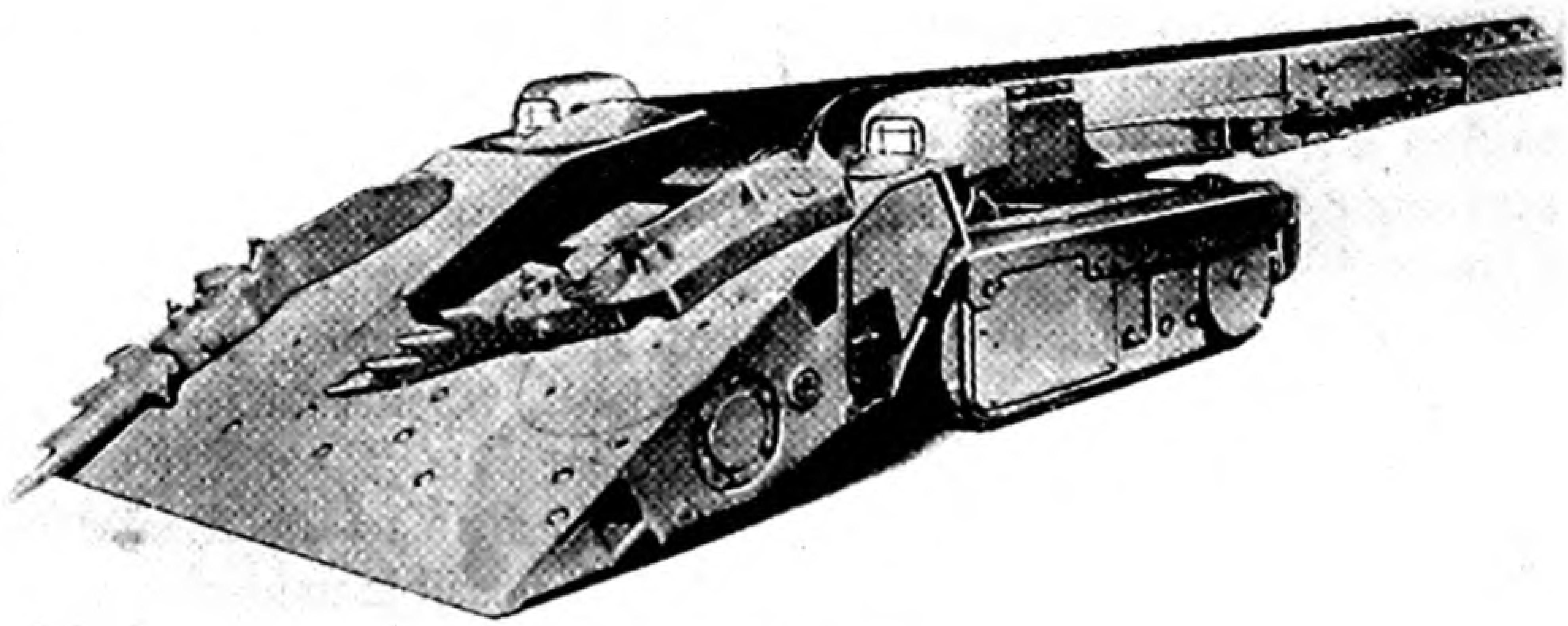


FIG. 138

pressed-air operation. The loader illustrated in Fig. 138 is structurally in four parts, and adjacent parts are joined by a hinge and two hydraulic jacks. The four parts are:

1. The crawler undercarriage, on which it is mounted.

2. The body, consisting of motor, reduction gearcase, transmission gear and gathering head, all rigidly bolted together. The body is hinged to the undercarriage by the crawler driving shaft housings. Jacks, placed between the gathering head and the front end of the undercarriage, can raise the gathering head by tilting the whole upper part of the loader about the hinge axis. This movement is used when flitting, or for lifting the head above steps in the floor.

3. The jib supporting frame, which is hinged to the gathering head, can be raised by hydraulic jacks mounted on each side of the machine.

4. The jib or rear part of the conveyor, which is pivoted to the

jib supporting frame, can be slewed laterally up to 45 degrees on either side of the centre-line by two horizontal jacks mounted on each side of the loader. The swing of the delivering jib is 7 feet 8 inches to each side, at the end of a 10 feet 10 inch radius.

The two gathering arms of the loader are flush-mounted on the ramp of the gathering head and the chain conveyor commences from a shallow hopper opening in the head and continues to the end of the flexible jib. Controls are conveniently placed on the right-hand side of the loader, and effected through multiple-plate and dog clutches. The hydraulic jacks for the movement of the jib and gathering head are controlled by piston valves. A 30-h.p. electric motor running at 1,500 r.p.m. is spigot-mounted on to the transmission gear case or, alternatively, a compressed-air turbine operating at 60 lb. per square inch may be fitted.

The leading details of the loader are as follows:

Width of machine	4 feet 10 inches.
Overall length	21 feet.
Minimum height	2 feet 8½ inches.
Height with delivery jib fully raised	4 feet 9½ inches.
Clearance, body to floor . .	3 inches.
Weight	6 tons.
Flittings speeds	54 and 155 feet per minute.
Loading capacity	4 tons per minute.
Crawler track width . . .	10 inches.

The loader may be operated to deliver on to scraper chain conveyors, belt conveyors, or directly into tubs or into shuttle cars. In operation, the loader is fed forward into the waste pile, the gathering head valve being held by a catch, so that the gathering head is kept to the floor by its own weight and follows any changes in floor level. The gathering arms pick up the stones, delivering them to the chain conveyor, which loads them on to the conveyor or tub behind the loader.

The 'Horzschaufel' loader. This machine forms a combination of a gate-end loader and a duck bill and is in use very successfully on the Continent, but it requires a good supply of compressed air.

The rocker shovel loader. This type of loader, which operates from a track, is designed to push into the heap of debris, scoop up the material in the shovel and, with an upward and over motion, dis-

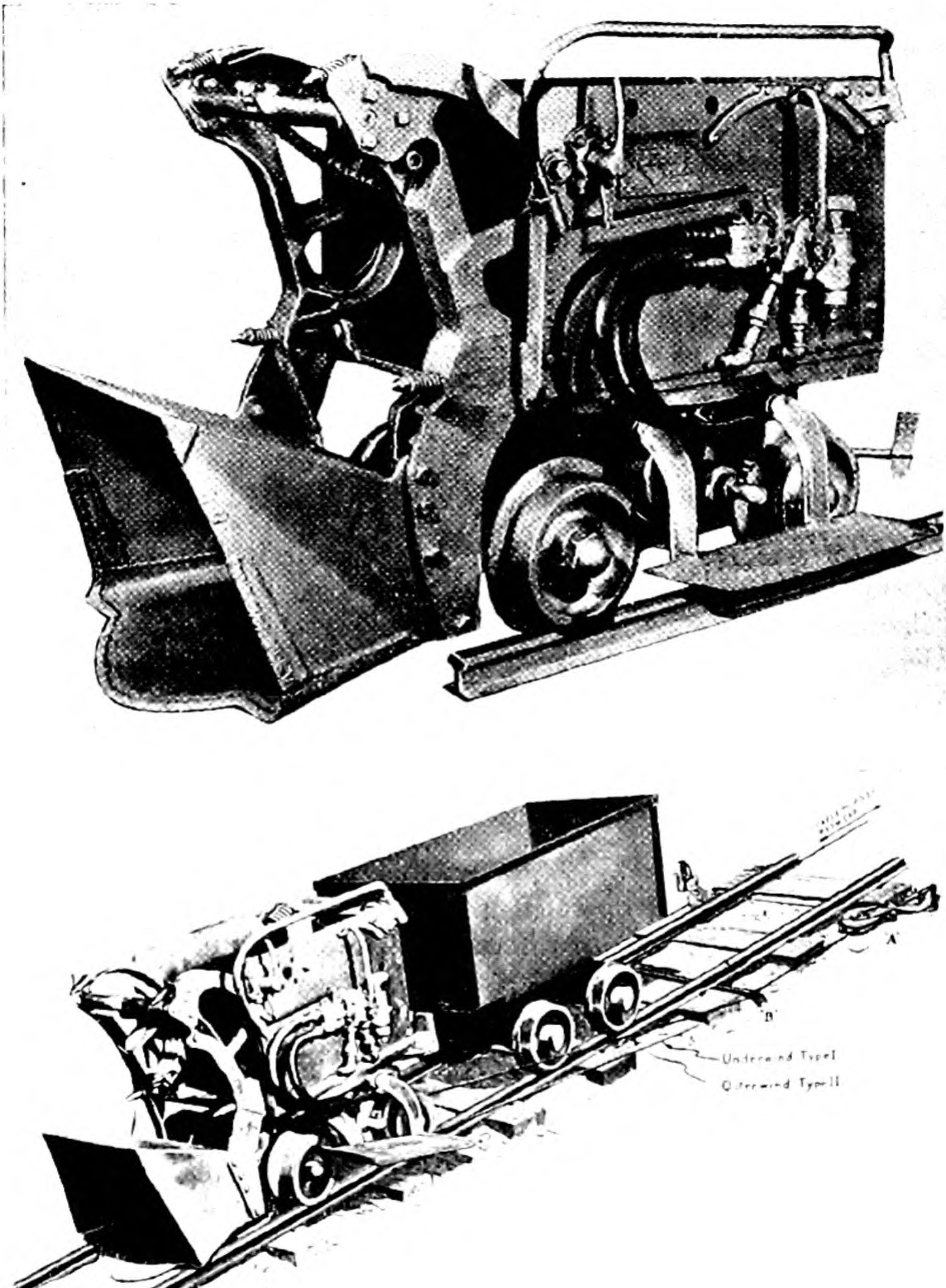


Fig. 139

charge the loaded bucket, or shovel, into a tub behind the loader. Several types of these loaders, which can be operated either by electricity or compressed air, are in use in British mines.

The Eimco-Finlay, Sullivan and Atlas Diesel and Salzgitter loaders are all of this type. Higher capacity shovel loaders, such as the Conway loader, are in use in large tunnel drivages. The Eimco-Finlay 12 B loader, illustrated in Fig. 139, is powered by two $5\frac{1}{2}$-h.p. piston-type air motors, one for loading and one for travelling. The operating lever controlling the movement of the loader and feeding into the bucket is placed conveniently for the left hand, while the control for digging, elevating, dumping and lowering the bucket is operated by the right hand. To feed the machine forward or backward, the left-hand lever is pushed forward or backward (this lever also controls the side-digging movements of the shovel). The right-hand lever is pushed forward to elevate the shovel, and if the lever is held in this position, the bucket completes its overturning cycle to the dumping position. The Salzgitter loader, which has a similar method of operation, incorporates an automatic tub coupling behind the machine. When this is released by the driver, it operates a compressed-air ram, which pushes the tub away from the loader and speeds up the tub-changing operation. The principal dimensions of the Eimco-Finlay 12 B loader are as follows:

Width of loader	$26\frac{1}{2}$ inches.
Overall length—bucket withdrawn .	42 inches.
Overall length—bucket extended .	76 inches.
Overall height—bucket lowered .	48 inches.
Headroom required	74 inches.
Discharge height of bucket . .	47 inches.
Discharge distance behind loader .	15 inches.
Clean-up range	76 inches.
Track gauge	$29\frac{1}{2}$ inches.
Loading capacity	25 tons per hour.

The loader is fitted to an undercarriage, which is self-propelled, and the shovel is capable of swinging through an angle of 30 degrees on both sides. Rock-loading machines fall into the classes which have been described. All have advantages and disadvantages, so that the choice of loading machine will depend upon the individual conditions of each drift. The following summary * indicates the

* *K.C.M.B.*, Vol. 2, Bull. No. 2, Series: Mech. No. 12—Drifting.

general features and compares the advantages and disadvantages of most types described:

Some Advantages of Slusher Loaders

1. The slusher can operate successfully over a considerable range of either adverse or favourable gradients. Gradients of 1 in 3 have been negotiated successfully.

2. It can be operated by either electric power or compressed air.

3. It is cheap and robust; maintenance costs are low per cubic yard of material mucked out.

4. When supply of tubs is interrupted, mucking can continue, the rock being left at the apron until tubs are again available. Mucking can continue while tubs are being changed under the loading jib.

5. The slusher is not in close proximity to the face during the mucking operation, and the operator is working under the safest conditions.

6. Operation of the slusher does not limit the width of heading.

7. Delivery on to conveyors can easily be arranged.

Some Disadvantages of Slusher Loaders

1. It is not mobile, and consequently must be left in the centre of the drift when not in use.

2. Track-mounted drilling rigs cannot be used. Speed of drilling may be limited by enforced use of post-mounted or hand-held drills.

3. All supports and other materials are man-handled from the slusher to the face of the heading or drift. Girder lifters cannot easily be employed.

4. Clean up of the muck pile is not complete. About 50 cubic feet of debris must be cleaned up by hand at the end of slushing operation and thrown back to be picked up by the skip.

5. Rate of advance is slowed on curves and bends.

6. There is a break in the cycle of operations when the slusher is moved up. Moving up and re-establishment in the new position may take a considerable time.

7. Loaded tubs cannot be propelled out from the face.

8. When the floor of the heading is soft, particular skill is required by the operator to prevent the skip digging in and leaving pot-holes (uneven floor).

9. The difficulty of fixing the sheave in soft rock.

Some Advantages of Rocker Shovels

1. The shovel can operate successfully on gradients positive or negative up to 1 in 10.

2. It is extremely mobile, small and compact per unit of loading capacity.

3. It can be used to propel and shunt empty and loaded tubs or cars at the face of the heading; tub-changing is a quick operation.

4. Access to face of heading is easy for supports and other supplies.

5. Track-mounted drilling rigs can be used.

6. Clean-up is very good over area covered.

7. Maintenance costs are not heavy per cubic foot of material loaded.

Some Disadvantages of Rocker Shovels

1. Rate of advance on heavy gradients is slow.

2. Only compressed-air power units are at present available.

3. Clean-up area is limited, and therefore width of heading is limited, unless two tracks are installed.

4. Rate of advance on bends or curves of self-centring models is slow unless designed for easy reversion to centring by hand.

Some Advantages of Gathering Arm Loaders

1. These machines are high-output machines and extremely mobile.

2. Clean-up is good, and possible area of clean-up does not limit width of heading.

3. Rate of advance is not slowed up on bends or curves.

4. It is adaptable to tubs or cars of varying dimensions.

5. Track-mounted drilling rigs can be used, though use is often limited.

Some Disadvantages of Gathering Arm Loaders

1. Maintenance costs are high.

2. Operation is more complicated.

3. Size/capacity ratio is low in comparison with other loaders.

In the survey of drifting installations carried out by Macaskill and Evans,* the rates of loading of several of the machines described are compared. The authors substantiate by the survey results their opinion that equipment having a high capital cost also has a correspondingly high loading rate, and that if the loading period only is considered, it is also more productive. To achieve the higher loading rate, extra labour must be employed for tub supply. Full utilisation of this labour throughout the cycle is necessary to retain a high overall productivity. The loading time studies taken in the survey are summarised in the following table:

Method	*Min. per cu. yd.*		*Approx. man-hours per cu. yd. (operational)*
	Range	*Av.*	
Slusher into tubs . . .	6·7 to 11·1	8·5	0·6
Eimco 21 into tubs . . .	6·6 to 7·2	6·8	0·5
8 B.U./4 into tubs . . .	2·1 to 7·7	5·8	0·6
Conway 6*a* into 25-cwt. tubs .	3·4 to 4·3	3·8	0·4
11 B.U. into 25-cwt. tubs . .	3·1 to 3·4	3·3	0·4
8 B.U./4 on to conveyor . .	3·2 to 3·8	3·5	0·4
11 B.U. on to conveyor . .	2·3 to 2·9	2·6	0·3

Section 3. Transport in Drifts

The rate of loading and the overall progress of the drift drivage is largely determined by the provision of adequate facilities for the supply of tubs to the face. The haulage arrangements should be capable of transporting debris with a minimum of effort and loss of time, so that the loading machine is utilised to its full capacity. Continuity in the loading operation can be achieved only by direct delivery of the debris from the machine on to a belt, chain or shaker conveyor. The chief disadvantages of such a system at the face of a drift in hard rock are as follows:

1. Damage to the conveyor during shot firing.
2. The necessity to extend the conveyor during loading.
3. The provision of special tensioning devices at the face for the belt conveyors.
4. A loading-point requiring periodic advancement has to be provided.
5. It may be necessary in the case of the slusher or scraper and

* *Proc. National Assoc. Coll. Managers*, 1949–50.

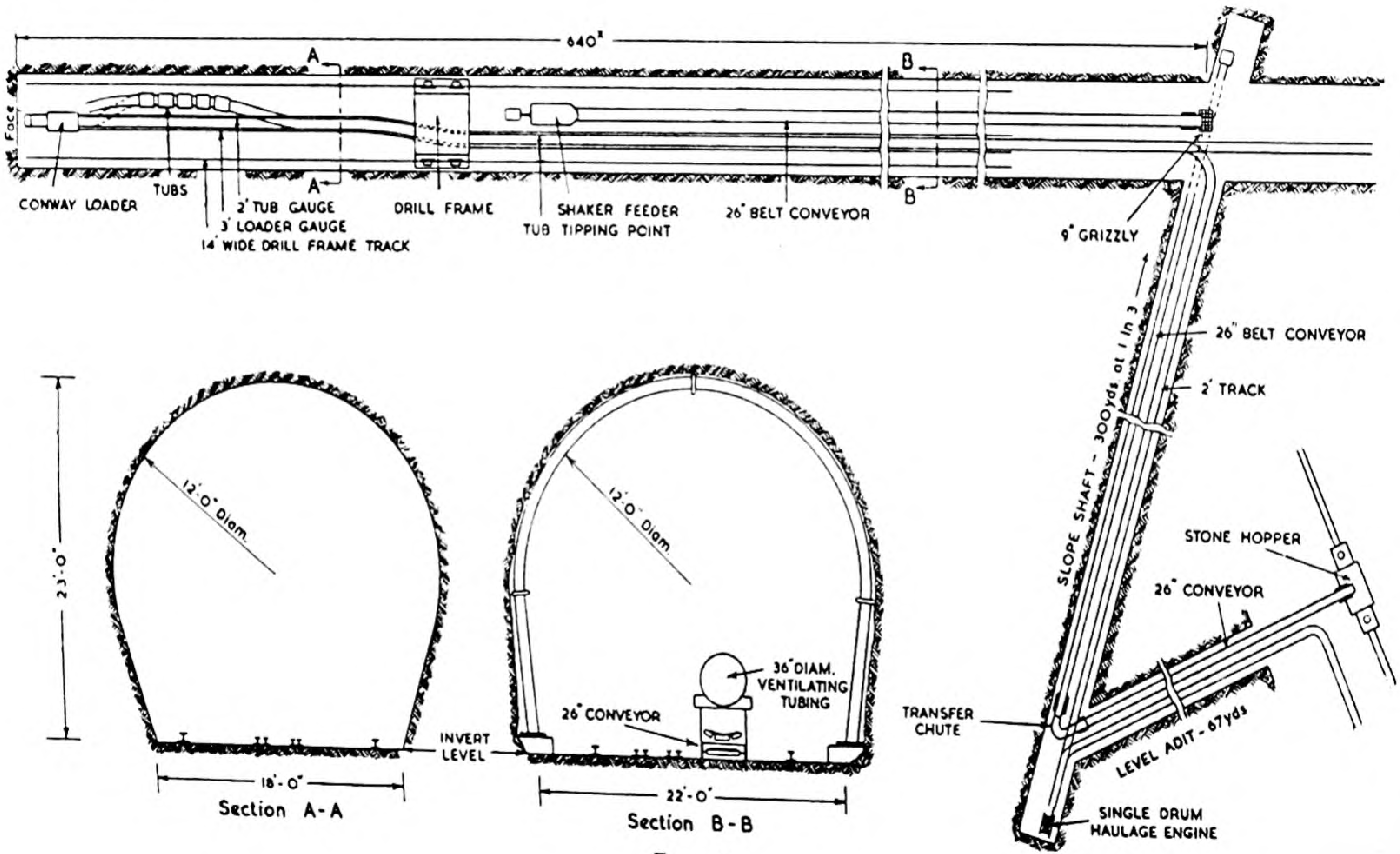
shovel loaders to modify the design of the machine or the layout to suit the conveyor used.

From these considerations, it is usual to find that direct loading into tubs is adopted. When loading from a single central track with a shovel or scraper loader, or on to a single track with a loader with gathering arms, the layout shown in Fig. 140 is a useful system. This drift, which utilised a Conway shovel loader, is arranged for debris disposal by a 26-inch belt conveyor. Six tubs are in use to convey the material from the loader to the tippling point on to a shaker feeder which delivers on to the belt conveyor. When a full tub has been hand-trammed from the face, a length of bonded track is swung into position to the centre loading track and an empty tub pushed under the loader. The empty tubs from the tipping point are run back to the empty loop for reloading. Such a bonded-track system is necessary, since in a drift advancing at, say, 30 yards per week, a laid crossing would have to be advanced more than once a week, or the maximum distance from the face to the tub changing-point would be as much as 40 yards. When loading from a double track with a shovel loader, a prefabricated crossing, as illustrated in Fig. 141, may be used. The full layout of the drift haulage system is illustrated in Fig. 142. A typical layout for the haulage system when adopting scraper-loading is shown in Fig. 143.

The haulage from the drift landing to the main haulage system, while drifting is in progress, may be by:

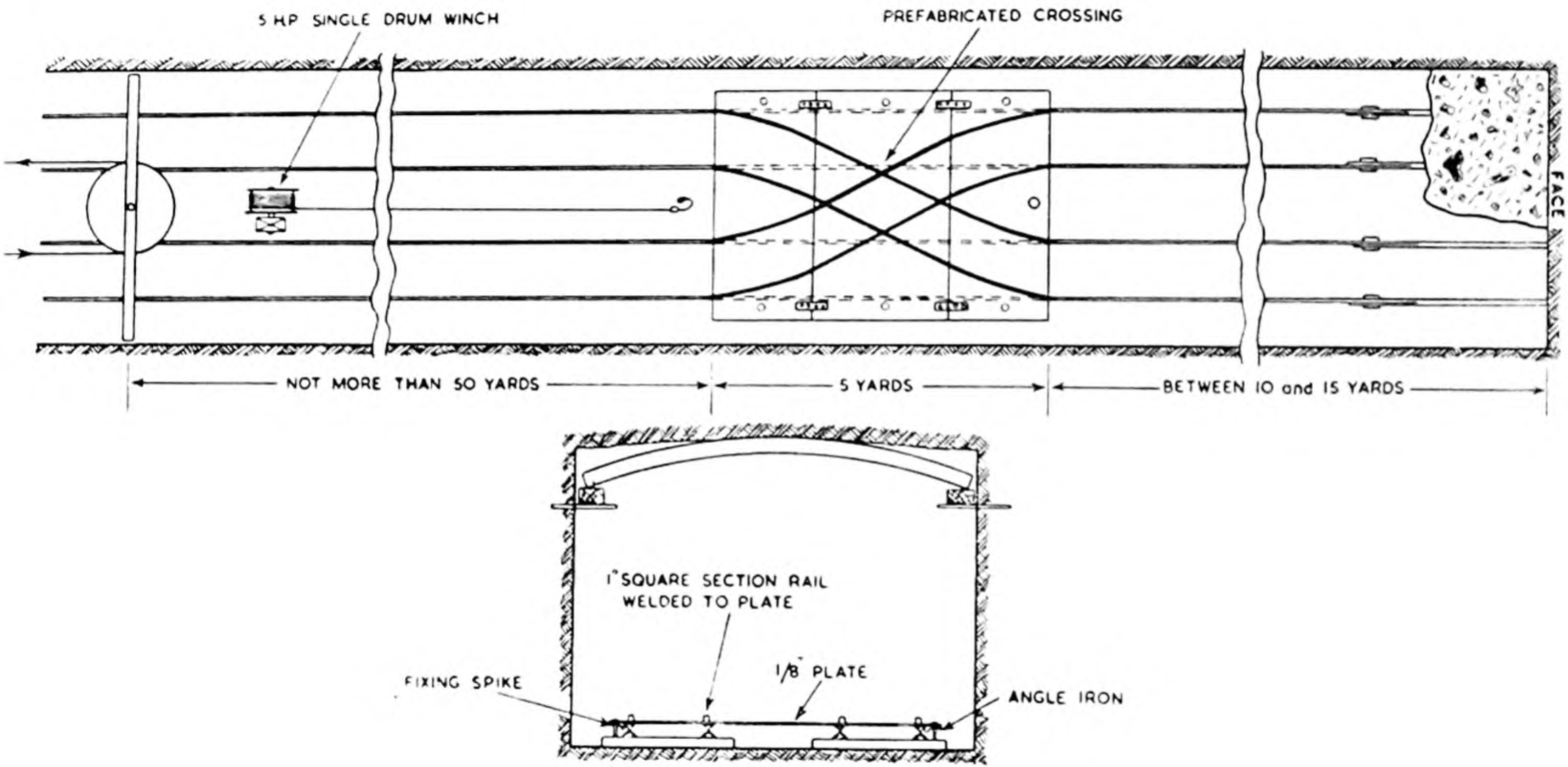
1. Endless or main and tail rope haulage.
2. Battery locomotive.

Although the battery locomotive is much more flexible in operation than a rope system, the choice would probably be limited to a rope haulage system. If the locomotive were to be utilised after the drift was completed, this system may be used. The cost of installation of battery-charging equipment, etc., for such an installation would normally be prohibitive. The use of diesel locomotives during drivage is prohibited except with special permission. Endless rope haulage is preferable to main and tail in a long drivage, being more flexible for the transport of tubs to and from the face, if used in conjunction with a small 5-h.p. winch situated at the return wheel. This arrangement is shown in Fig. 144, and full tubs are hauled to the return wheel by the winch.



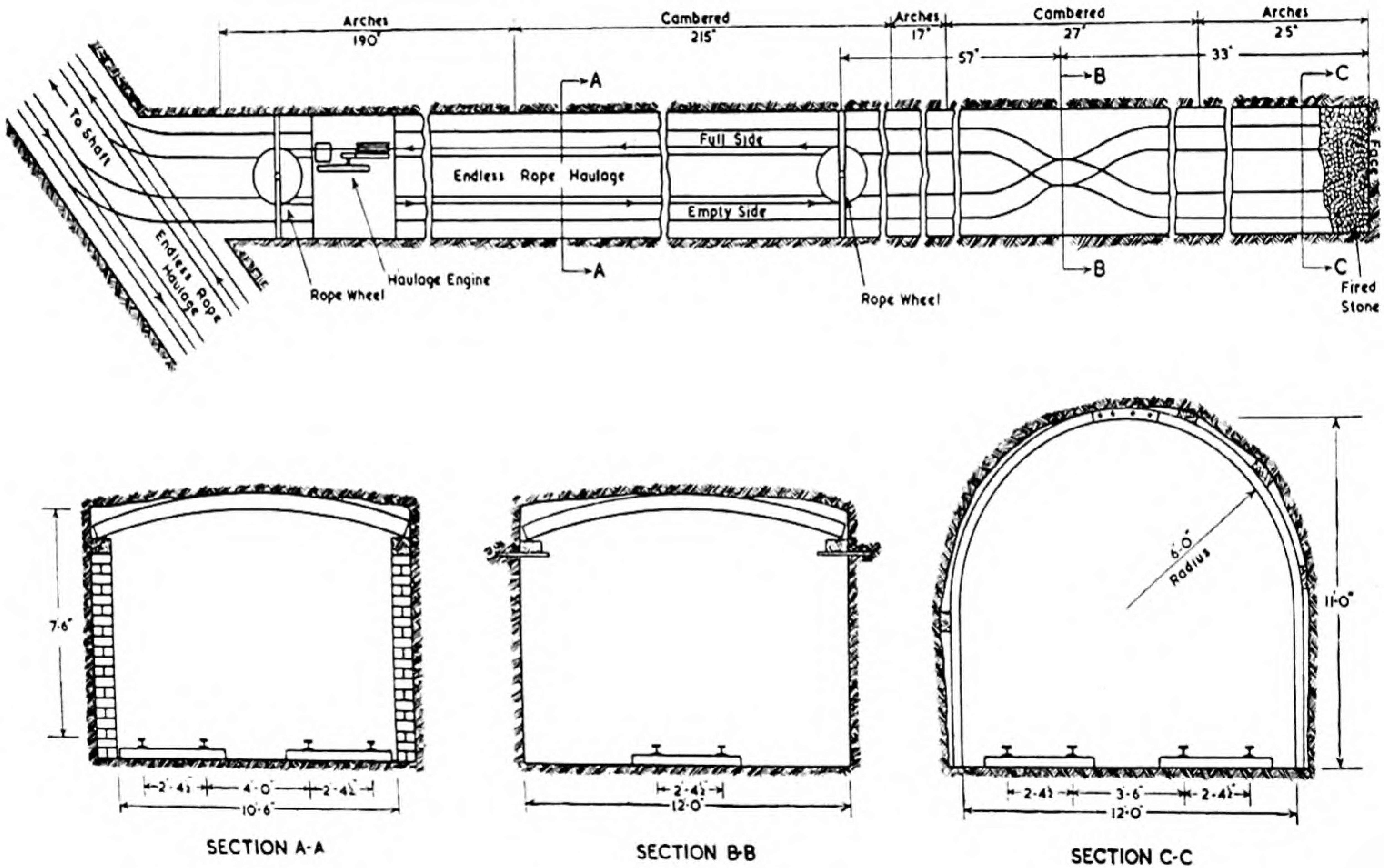
640'
Face
A
A
B
B
CONWAY LOADER
TUBS
2' TUB GAUGE
3' LOADER GAUGE
14' WIDE DRILL FRAME TRACK
DRILL FRAME
SHAKER FEEDER
TUB TIPPING POINT
26" BELT CONVEYOR
9" GRIZZLY
SLOPE SHAFT - 300yds at 1 in 3
26" BELT CONVEYOR
2' TRACK
STONE HOPPER
26" CONVEYOR
TRANSFER CHUTE
LEVEL ADIT - 67yds
SINGLE DRUM HAULAGE ENGINE
23'-0"
12'-0" Diam.
12'-0" Diam.
36" DIAM. VENTILATING TUBING
26" CONVEYOR
INVERT LEVEL
18'-0"
Section A-A
22'-0"
Section B-B

FIG. 140



— SECTION AT PREFABRICATED CROSSING —

FIG. 141



Arches
190'
Cambered
215'
Arches
17'
Cambered
27'
Arches
25'
57'
33'
A
B
C
To Shaft
Endless Rope Haulage
Rope Wheel
Haulage Engine
Endless Rope Haulage
Full Side
Empty Side
Rope Wheel
Face
Fired Stone
7·6"
2'·4½"
4·0"
2·4½"
10·6"
SECTION A-A
2·4½"
12·0"
SECTION B-B
6·0"
Radius
11·0"
2·4½"
3·6"
2·4½"
12·0"
SECTION C-C

FIG. 142

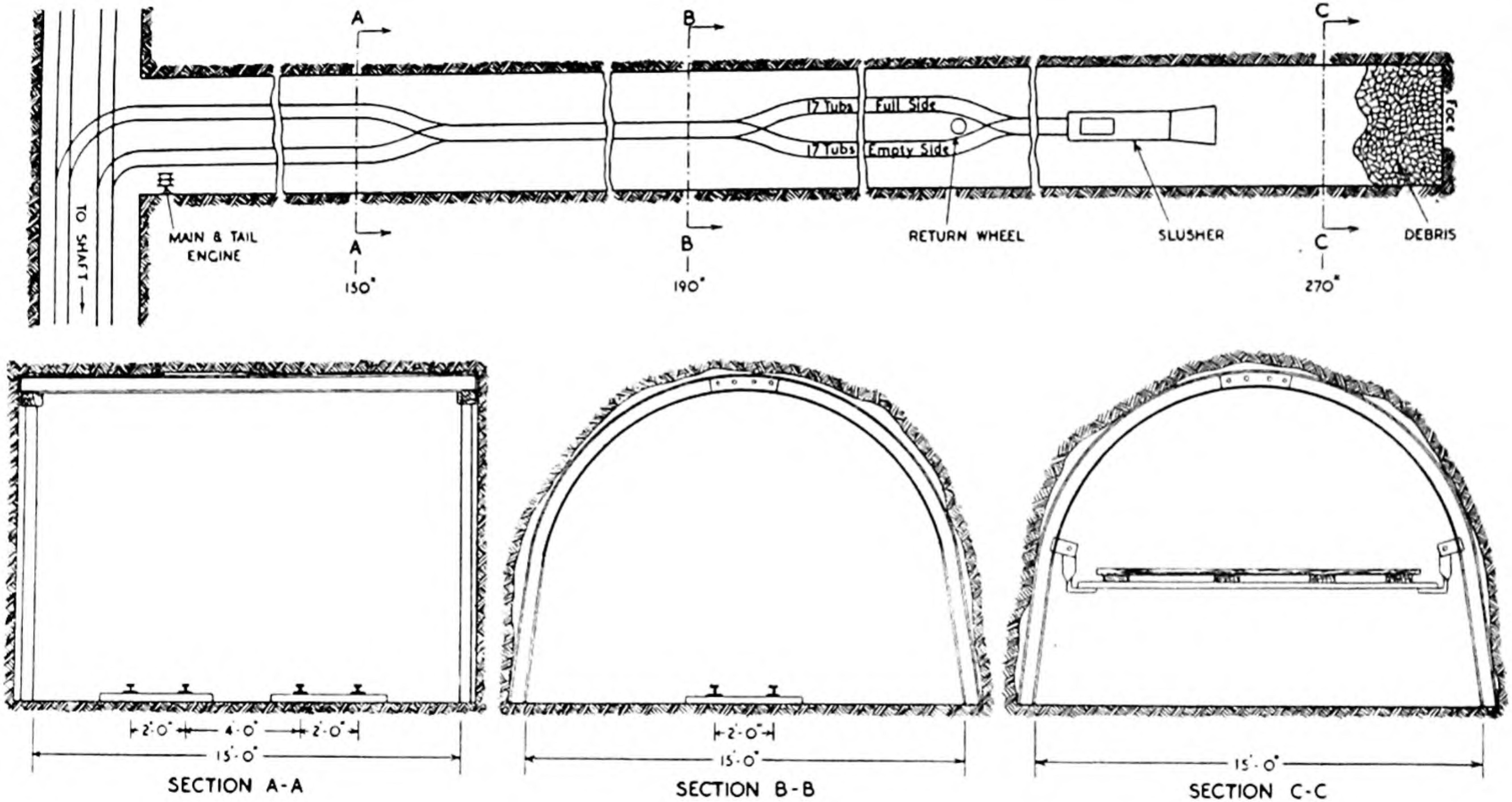

TO SHAFT
MAIN & TAIL ENGINE
A
A
150'
B
B
190'
17 Tubs
Full Side
17 Tubs
Empty Side
RETURN WHEEL
SLUSHER
C
C
270'
DEBRIS
Face
2'·0"
4'·0"
2'·0"
15'·0"
SECTION A-A
2'·0"
15'·0"
SECTION B-B
15'·0"
SECTION C-C

FIG. 143

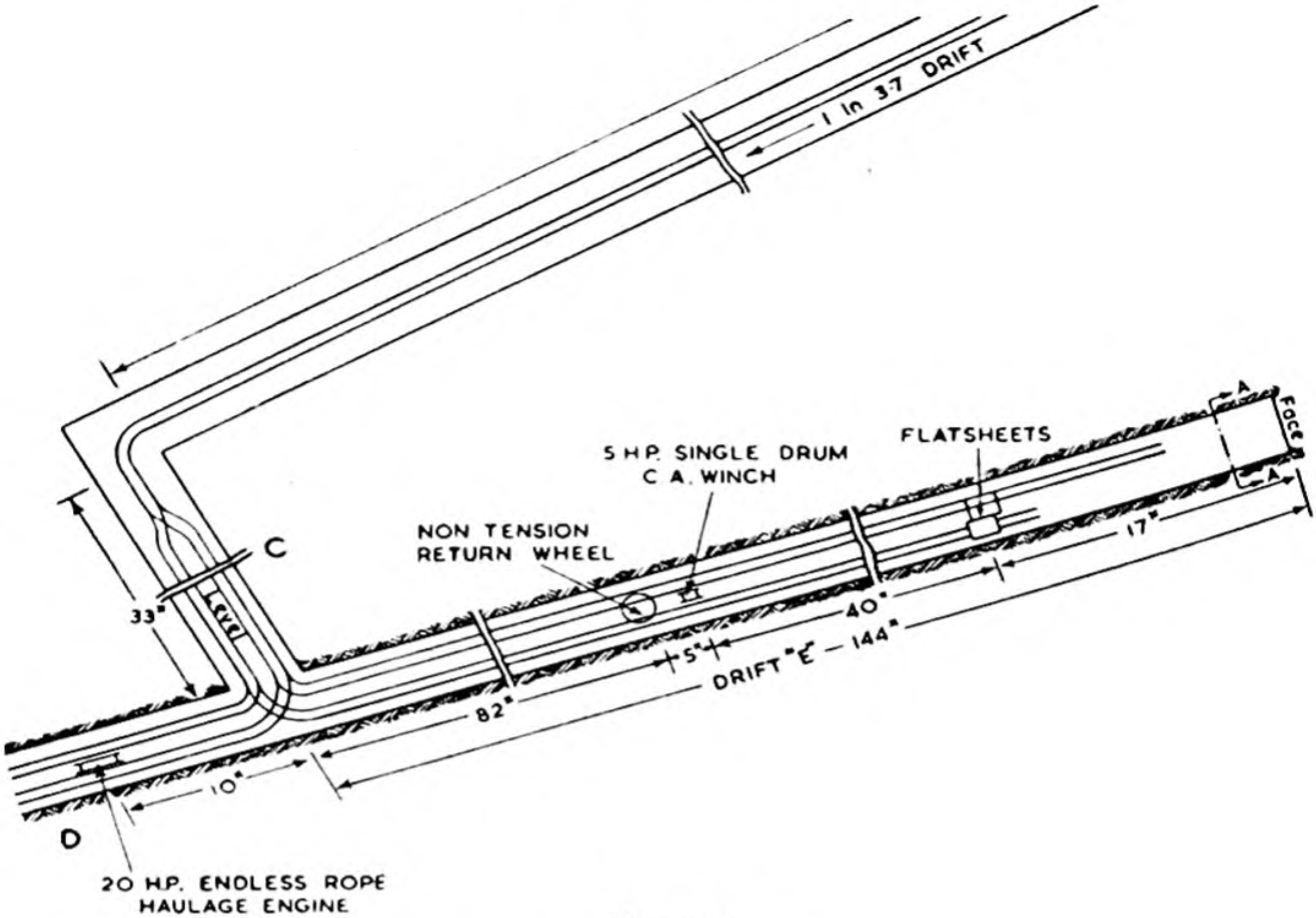


Fig. 144

Section 4. Organisation

(*a*) *Introduction.* In the efficient organisation of stone-drifting development, the important factor is the necessity to conduct each individual operation, both technically and economically, to the best possible advantage. In order to reduce wasted time and effort to the minimum, each operation is organised to follow in its appropriate place in the complete cycle of operations. The correct number of men should be employed on the work, as both under- and over-estimating the labour requirements produce poor results. The greater the rate of advance, and the more expensive the equipment for drilling and loading, the greater is the importance of capacity working and full utilisation of both labour and machines.

(*b*) *Hand loading stone drifts.* Where the drift is being loaded out by hand, the usual cross-sectional area is from 95 to 150 square feet and the performance per manshift varies between 1·3 and 2·3 cubic yards. The performance can be expressed also in terms of inches per manshift, 3 inches per manshift being a low rate of performance and 6 inches per manshift a high performance. The variations in per-

formance in the first instance will be governed by the number of men and their distribution in the individual shifts worked, whether the drifting is in progress over single, double or three shifts. A large time-consuming factor is normally drilling; the speed with which this can be done depends upon the type of rock to be drilled and fired, and whether ordinary hard steel or carbide bits are being used as well as the type of drilling rig being employed. The type of explosive required is also a major factor, since this will influence the total number of holes required and the number in each round. The nature of the strata will decide the matter of supports, whether these are required and, if so, how much stripping is needed before these are brought to the face and set. The haulage arrangements installed will influence the quick turn-round of tubs or mine cars and, together with other factors, such as water feeders and the working temperature, will affect the general progress of the drifting schedule. One major factor in the efficient organisation of the drifting is the fixing of a piece-work contract on a basis agreeable to the men, thus providing an incentive to them to complete the work quickly and efficiently.

The number of men employed in a drifting team will depend on the size of the drift to be driven as well as on whether the work of individual operations can overlap. Drilling and bringing in supports can be conducted at the same time; drilling and loading can be conducted simultaneously; or drilling can be done while setting supports and laying track.

Normally the minimum number of men employed at the same time is three, who are capable of drifting in sections up to 100 square feet; for sections between 100 and 130 square feet, four or five men are employed; and for more than 130 square feet, a team of five men is required. Considering a drift with a section of 135 square feet with a pull per round of 2·2 yards, the excavation produced is 33 cubic yards. If a performance of 1·3 cubic yards of solid rock per manshift is obtained, which may be the case in sandstone, twenty-five shifts will be required for an advance of 2·2 yards. If this lift is taken in one day, twenty-five men have to be employed. Where the drift is organised in three shifts, from twelve to fifteen men can be usefully employed, while with four shifts in operation (considering the actual face shifts and excluding other time taken up by travelling and allowing an overlap) from eighteen to twenty men

can be employed. This implies a performance of 1·5 yards, or at the most 1·7 yards, per day which, in sandstone rock and with hand-loading, should be regarded as satisfactory.

If the drift is being driven in softer measures, such as a soft shale, fewer holes will be required and drilling and charging will take up less of the total time. The performance per manshift with hand-loading can be increased from 1·7 to 2·0 cubic yards of solid rock, and with post-mounted machines (such as the air-leg), even 2·5 cubic yards of solid rock is possible. This performance will give an excavation of 37·5 cubic yards, or at a cross-sectional area of 150 square feet, an advance of 2·25 yards, per day with fifteen men, five in each of three shifts.

The drifting work should be organised as far as possible so that a round is lifted per day. With shale ground and using post-mounted drills with gelignite for firing down the round, this target is feasible, although a four-team organisation may be required.

In order to illustrate the sequence of operations and the possibility of attaining this performance, the following example is considered:

The drift is a lateral drift, 150 square feet in area, being driven in sandy shale and shale ground. The supports consist of steel circular rings set 2 feet 6 inches apart; thirty-five holes are required (four cutters, six sump holes, fourteen easers and trimmers with six bottom lifters and five roof holes). The explosive used is gelignite, the thirty-five holes being fired simultaneously in three rounds, cutters first, followed by sumpers, easers and trimmers, and then the bottom lifters and roof shots.

The cycle of operations is as follows:

1st Shift. 6 a.m.–12 noon	2nd Shift. 12 noon–6 p.m.
5 *men* drilling about 30 holes with 4–5 hammer drills.	4 *men* drilling remaining holes. Fire first two rounds and load out.
3rd Shift. 6 p.m.–12 midnight	4th Shift. 12 midnight–6 a.m.
5 *men* complete loading out. Fire last round and load out.	4 *men* set supports, lay track and lengthen air and vetilation ranges.

The number of men employed per day is eighteen and the daily advance to the length of the pull is 2·2 yards. The performance per manshift is 2 cubic yards, or 4·4 inches. Using post-mounted machines mounted on one or two columns and operating the four or

five machines with three men, the performance can be increased to 2·3 cubic yards, or 5 inches per manshift.

(*c*) *High-speed drifting with mechanical loading*. Using mechanical loading methods, performances per manshift of from 2·6 to 4·8 cubic yards, or from 6 inches to 12 inches, are being obtained.

These performances are being obtained with from two to three pulls, or from 4·5 to 7 yards advance per day. Such performances are based not only on the decrease in the overall loading time by the use of mechanical loading equipment but, also, on the speeding up of the drilling cycle, the good maintenance of the equipment, a constant supply of compressed air at the required pressure and the efficient organisation of the cycle of operations involved. Continuous cyclic working over twenty-four hours with the optimum number of men, fully employed, must be arrived at over three or four shifts. With a four-shift working cycle, the normal overall shift time is reduced to include the maximum time at the face. The shorter shift promotes better working conditions for the men and allows a continuous cycle without a break between shifts or the staggering of the individual shift times of the operators, which is required with the three-shift working if continuous operation is desired.

When considering the actual organisation and distribution of labour, an eight-hour cycle should be possible in three shifts or a six- or twelve-hour cycle in four shifts. The advance per day depends on the time necessary for a cycle and on the advance per lift. With deeper lifts considerable advantages will be gained, such as considerable saving of time in preparation work and additional work, which is much the same for each length of a lift. Therefore deep lifts of 3 to 3·5 yards are more efficient than short lifts of about 2 yards. A good performance will be 3 lifts of 2·2 yards or 2 lifts of 3·3 yards per day. With mechanical loading and drill rigs, with good organisation 4 lifts of 2·2 yards or 3 lifts of 3·3 yards are also within reach. Practical application in the individual circumstances and conditions will decide which of these systems can be adopted. Where it is possible to attain the high performance of 3 yards or more in an eight-hour cycle working three shifts, this system should be the final aim. The deciding factor is the distribution of the total time among the various operations involved. The operational times can be kept within the following ranges:

	Hours
1. Drilling	1 – $2\frac{1}{2}$
2. Firing	$\frac{3}{4}$ – 1
3. Stripping down and setting temporary supports	$\frac{1}{2}$ – 1
4. Loading	1 – 3
5. Setting permanent supports, advancing track and lengthening compressed-air range, etc.	$1\frac{1}{2}$ – 3

It may be possible to provide a certain flexibility in the individual operational times by setting supports, conducting outbye work and drilling simultaneously. The overlapping of the drilling and charging operations will not normally be practicable. Experience has indicated that the above operational times are sufficient to cover pulls of about 2·2 yards. The initial difficulties which usually arise will restrict the lifts to smaller proportions, such as 1 to 2 yards advance, so that the cycle of eight or twelve hours can be maintained. The reduction of the length of the lift allows sufficient time to compensate for lost time and to restore the normal shift cycle within a reasonably short period. The best performance per man-shift can be reached only if the longest possible lift is obtained. Each operation must be planned carefully and carried out efficiently. The following points should be given close consideration:

In the case of stone drifts being driven with mechanised loading equipment, drilling is the most critical operation for disturbing the rhythm of the cycle. The choice of the most suitable equipment and its maintenance is important, especially the drilling bits, and more particularly if they are of the carbide-tipped type. Depending upon the rock hardness, drilling performances of from 5 to 40 inches per minute can be achieved with carbide bits, inclusive of supplementary time taken up for drill-changing and positioning of the machine. Depending upon the size of the drift being driven, the hardness of the rock and the type of explosive being used, from 200 to 400 feet of drilling will have to be done per round. Under the most favourable circumstances a drilling time of 100 minutes will be required, and this may be increased under adverse conditions to 1,000 minutes. The average drilling cycle will take about 400 minutes. In order to keep the total drilling time within the range of from 1 to $2\frac{1}{2}$ hours (60 to 150 minutes), it is necessary to operate a maximum number of drills simultaneously. An optimum of four drills, using a drilling car or rig, is regarded as satisfactory, although

more can be used, but with a probable reduction in the performance per manshift. When using heavy rotary drills, two drills will be sufficient.

The use of ordinary compressed-air-operated drill mountings, such as the air-leg, has been found satisfactory, but rigs with transverse post mountings are recommended, providing that the setting and dismantling time is not greater than from 15 to 30 minutes. Any increase on this time cannot be made up in a total drilling time of from 1 to 2½ hours. The same comment can be applied to special drilling trucks or bogeys, and this is one of the main reasons for their limited application.

The air pressure required at the face for drilling is extremely important and this should be at least 80 lb. per square inch; very good results have been obtained with pressures up to 120 lb. per square inch, which may be provided by a booster compressor. Since the force of the blow of a percussive machine is dependent upon the pressure of the air at the machine, an efficient compressed-air system and supply is essential.

The simultaneous firing of the sequence of holes in the round is imperative for a high-speed drifting cycle, since sufficient rock must be broken down to employ the loading machines at an efficient rate. The use of low-tension as against high-tension detonators is necessary, therefore, since series firing can be carried out. The higher consumption of explosives required, which may amount to 20 per cent., must be considered.

The avoidance of misfires is essential in order to continue the cycle without disturbing delays. Efficient detonators and simultaneous shot-firing equipment, such as the exploder and circuit tester, are important items as well as the care and supervision of the proper insulation of primer wires and cables. The charging operation should be carried out speedily and carefully.

The work required to strip the roof and sides after firing a round cannot be reduced except by the provision of proper equipment for ease and speed in doing the work.

The setting of permanent supports and lengthening of track should be combined with the drilling work if at all possible. The greater the difficulty in setting supports, the more necessary it is to reduce the drilling, charging and firing times in the cycle. In highly mechanised drifts, setting supports is often the most time-consuming

factor. To overcome this, with good roof conditions, temporary supports are sometimes used, which are replaced by a special team following the face at some distance. Where lengthening of the track is retarded, due to lack of time, the use of a temporary set at the face is recommended, leaving the final track-laying to another team. During drifting under normal conditions, the type of loading machine is less important than the efficient running of the machine during the loading operation and within the framework of the complete cycle of operations. A good operator is necessary and his choice and pretraining in the operation of the machine essential. With compressed-air-operated loading machines, the air pressure at the machine is also of major importance.

The necessity for good supervision and inspection of the work is obvious, and a good team spirit among the operators is a first essential. The men should be given an incentive to do the work well by setting a proper contract for the work and retaining their interest in the job to be done.

Daily reports are required for the efficient control of the individual operations, while the progress and performance of the drift should be recorded in monthly summaries. A form of daily report on the supervision of the work is shown in the table on the opposite page.

(*d*) *Comparison of costs on hand loading and mechanically loading drifts.* Since high-speed drifting with mechanised loading equipment introduces a heavy initial capital cost, as against the normal development of a level by hand-loading methods, it can be justified only by an increase in efficiency and a reduction in the wages and other relevant costs. Working conditions are also an important factor, since hand-loading entails deep breathing and is apt to promote silicosis. If the drift to be driven is required urgently in order to commence a new level or working panel, the additional cost may be justified, especially at a new mine where the start of actual coal production is required quickly to offset the development and equipment costs and reduce charges on capital expenditure. The decision is also influenced by insufficient labour being available to conduct the work by ordinary methods. Generally, hand-loading methods are the more expensive, since the performance per manshift which can be achieved is limited. A lower cost can be reached using mechanical loading if a complete round is loaded per lift. If the advance is increased to more than one round lifted per day, there is

Supervision form for mechanised stone drifting

Mine: horizon: section of mine: road: *cross-cut cross-section unlined:* 150 *sq. ft.*

loader: supports: *steel arches with Moll joints* cross-section lined: 120 *sq. ft.* spaced rock: *sandstone.*

		Drilling			Support			Blasting			Filling			Additional Work					Advance		Number of shifts		Performance per man-shift				
																							Length		Room excavated		
day	shift	time	number of holes	total length of holes	time	number of sets	lagging	time	normal time	additional time	time	number of tubs	performance	time	track	pipe-line	vent-tubes	water ditch	per day	total	per day	total	per day ins.	total ins.	per day cu. ft.	total cu. ft.	Remarks
		hrs.		feet	hrs.		sq. ft.	hrs.	hrs.	hrs.	hrs.		tubs per hr.	hrs.	ft.	ft.	ft.	ft.	ft.	ft.			ms.	ms.	ms.	ms.	
a	*b*	*c*	*d*	*e*	*f*	*g*	*h*	*i*	*k*	*l*	*m*	*n*	*o*	*p*	*q*	*r*	*s*	*t*	*u*	*v*	*w*	*x*	*A*	*B*	*C*	*D*	*E*
1	1	3	30	250	6	1	150	1¼	1¼	—	1	5	5	1	17	17	14	14	—	—	6	—	—	—	—	—	—
	2	—	—	—	6	1	150	—	—	—	6	68	11	—	—	—	—	—	—	—	6	—	—	—	—	—	—
	3	3½	30	250	—	—	—	1½	1½	—	—	—	—	—	—	—	—	—	—	—	4	—	—	—	—	—	—
	4	—	—	—	—	—	—	—	—	—	5	60	12	—	—	—	—	—	14	—	4	20	8	8	105	105	—
2	1	4	34	290	6	1	150	1½	1½	—	1	10	10	1	—	17	14	14	—	—	6	—	—	—	—	—	—
	2	—	—	—	6	1	150	—	—	—	5	60	12	1	17	—	—	—	—	—	6	—	—	—	—	—	—
	3	3½	30	250	—	—	—	1½	1½	—	1	14	14	—	—	—	—	—	—	—	4	—	—	—	—	—	—
	4	—	—	—	—	—	—	—	—	—	5	74	14	1	17	—	—	—	14	28	4	40	8	8	105	105	—

an increase in the total cost, but this should not exceed 5 per cent. of the minimum cost of one lift per day.

The cost of supports, track and pipes remains constant and is independent of the advance per day. The drilling cost is dependent upon the type of bit and machine used, increasing with the more expensive alloy bit. The same comment can be applied to the explosives cost. The mucking cost is increased if mechanical loading is adopted, but this is offset by the increased daily advance as the rate of utilisation of the machine is increased. The wages cost is reduced considerably by the use of mechanical loading and alloy bits. If the advance is increased from one to two or three lifts per day, the costs will remain almost static or show a slight increase. An increase in cost will occur when the number of operators per shift must be high to maintain the rhythmic cycle of the work and, in such cases, a certain amount of individual interference cannot be avoided. Auxiliary ventilation costs will decrease with increasing rate of advance.

(*e*) *Examples of high-speed drifting organisation.* The previous conception of high-speed drifting organisation can be illustrated by examples which show the number of persons employed and the distribution of the work when different types of loading equipment are used.

Case 1. A scraper loader is being used in a level drift of 145 square feet cross-section. The drift is being driven half in sandstone and half in shale. Drilling is being carried out, using carbide bits and

6 a.m.–12 noon	12 noon–6 p.m.
3 men with 3 hammer drills drill 36 holes. (269 feet.) 3 men set the 4 props of the supports, lengthen track, advance air line and do other additional work. Firing.	5 men drill 4 roof holes and finish setting supports. Load 75 cars using scraper loader.
6 p.m.–12 midnight	12 midnight–6 a.m.
3 men with 3 hammer drills drill 36 holes. (269 feet.) 2 men set 4 props for 2 girder sets and lengthen track and advance air line. Firing.	5 men drill 4 roof holes and finish supports. Load 75 cars using scraper loader.

firing with gelignite in the solid rock or using sheathed explosives near the seam contacts. One round requires forty holes. The roof supports are polygonal steel arches or wooden props. The rate of advance is two lifts of 2·3 yards each, giving a daily advance of 4·6

yards. The performance corresponds to 3·5 cubic yards., or 7·9 inches, per manshift. A team of twenty-one men are employed, the drifting operating on a twelve-hour cycle distributed over four shifts.

Case 2. A shovel loader (Salzgitter or Eimco type) is being used in a cross-cut with a sectional area of 140 square feet. This requires forty 7-foot holes per round. Gelignite explosive is being used. The supports are the doorstead type. The capacity of the loader is 16 cubic yards of debris. The temperature at the face is 32° C., and the actual working time is five and a half hours. The drift is being driven by a twenty-four-man team divided over four shifts and working to a cycle of twelve hours. The performance per manshift is 2·9 cubic yards of solid rock, or 6·7 inches.

6 a.m.–12 noon	12 noon–6 p.m.
5 men drill and fire 40 holes. 3 men set supports.	3 men loading out 60 cubic yards. 1 man digs out drain.
6 p.m.–12 midnight	**12 midnight—6 a.m.**
5 men drill and fire 40 holes. 3 men advance track and air line.	3 men load out 60 cubic yards. 1 man digs out drain.

Case 3. The loading machine being used is a reciprocating shovel loader. Of the face men, four are on four shifts working on a piece-work rate, while four timber-men and one fitter are on the morning shift at a shift rate. The advance is two pulls per day, 2·2 yards each, corresponding to a twelve-hour drifting cycle over four shifts. The actual working time at the face is six and a half hours out of a shift time of eight hours. The individual shifts overlap at the beginning and end of the shifts. The overlapping time is utilised on back-bye work in order to reduce interference with the individual operators. The performance per manshift is 3·5 cubic yards., or 7·5 inches.

6 a.m.–2 p.m.	12 noon–8 p.m.
4 men load out. 2 men set 2 supports, advance track and air line. 1 fitter.	4 men with 3 hammer drills drill 40 holes, fire and strip down face for loading. Advance face horse-head supports.
6 p.m.–2 a.m.	**12 midnight–8 a.m.**
4 men load out and set the support.	4 men with 3 hammer drills drill 40 holes, fire and strip face for loading. Advance face horse-head supports.

PART II

SUPPORT OF MAIN ROADWAYS

Section 1. *Stress Distribution around Stone Drifts—Static Stress*

The change in the original stress distribution in the strata layers resulting from the development of working faces in the seams, discussed previously, is reproduced by the drivage of roadways in the rock. Considering a particular particle of rock, it is subjected to a pressure due to the gravitational weight above it which is counteracted by a reaction from below. Because the stone particles cannot expand themselves in a lateral direction, they are also suffering lateral pressure, which, however, before any workings are made in the neighbourhood, will be much smaller than the vertical load. Thus the layers are subjected to an all-round but unequal compressive stress. The intensity of this compressive force at any point caused by the weight of the overlying beds is defined as the static pressure, and it increases in proportion to the depth. The equilibrium conditions existing before the road was driven are altered due to the immediate upper beds being unable to support the overlying strata above the excavation. Thus the weight of the upper beds or the static pressure is transferred to each side of the roadway, which has been adequately proved, by experience, calculation and investigations on models. The roadway sides are in zones of increased compressive stress, while the floor and roof are in zones of tensile stress. Since the original stress distribution before drivage of the roadway can be represented by parallel vertical lines (the considerably smaller lateral compressive stress, amounting to from one-tenth to one-quarter of the vertical stress, can be neglected in this instance), these lines of stress will be deflected away from the excavation and be concentrated in the solid at the roadway sides. The change in the distribution of vertical stress takes the form of an ellipse, called the pressure ellipse. It is in the form of this elliptical zone that the stresses will increase and change. This assumption is shown to be true by observations underground in roadway rippings, which tend to correspond to the upper half of the ellipse represented by the deflected lines of stress.

These factors can be discussed under the separate headings:

(*a*) *The effect of the static pressure on the strata.* The effect of the

vertical static load on the strata above an excavation depends upon three main factors, viz. the strength of the rock, the depth and the shape of the roadway cross-section.

If the compressive and tensile strengths of the rock are greater than the compressive and tensile stresses to which it is subjected, no effects will be observed. Taking the compressive strength of sandstone as 14,000 lb. per square inch, and the tensile strength as 450 lb. per square inch, at a depth of 1,400 feet, the rock is under a pressure of about 1,400 lb. per square inch. The increase in the compressive stress at the sides of the roadway may be about three times the static load, or 4,200 lb. per square inch, which value is well below the ultimate strength of the rock. If the tensile stress remains less than 450 lb. per square inch, no effect will be observed on the surrounding strata. If, however, the rock is loaded to more than its ultimate tensile strength, the roof will begin to break, forming fissures at the roadway sides in the zones of high tensile stress until finally the roof collapses from the sides. Such conditions would arise if the rock had a compressive strength of 7,000 lb. per square inch, at a depth of 2,800 feet. The original static pressure at this depth is 2,800 lb. per square inch, and assuming a threefold increase in the side pressure to 8,400 lb. per square inch, which is very much higher than the ultimate tensile strength of the rock, the roadway sides fail due to shear and the roof caves in. In the case of an extremely strong rock at moderate depth under static pressure, supports may not be required, while supports are necessary with less solid rock strata or when driving in strong rock at greater depths.

Sandstone is approximately elastic down to a depth of 1,000 yards and is brittle, whereas shale, which is considerably less solid than sandstone, becomes plastic even at moderate depths and undergoes complete change in form when subjected to small pressures. When the roadway is driven in shale, the vertical static pressure will cause the roadway sides to bulge; where the shale forms the roof, the roof will bend, and where it forms the floor, 'creep' occurs due to uplift of the floor. Uplift of the floor can also be caused by the additional pressure on the roadway sides where the floor is weak and the roof strong. Generally, roadways driven in shale require stronger and a more continuous form of support than is required in sandstone, sandy-shale taking a mid-way point between these extremes.

(*b*) *Influence of the cross-section of the roadway upon the effects of strata pressure.* The size and shape of the roadway is extremely important when considering the effects of strata pressure. The larger the roadway cross-section, the greater the resulting effects will be, and this is especially so for tensile roof stresses.

The influence of the cross-section can be seen without difficulty from the pressure ellipse. This ellipse is, to a certain extent, the form which the surrounding rock under static pressure conditions is trying to force upon the roadway cross-section. This ellipse is relatively high and narrow, in the case of the roadways, so that high roadways are more favourable than low roadways of greater width. These suppositions have been confirmed both by practical investigation and by tests on models using a photo-elastic technique, which tests have confirmed that the trapezoidal and rectangular sections are the most unfavourable form to withstand the pressure conditions. The roadway sides undergo heavy compressive stress, which can be seen by the heavy concentration of lines of principal stress in Fig. 145A. Quite apart from this, the tensile stresses are also concentrated in the floor and in the roof layers. Where the section is semicircular or polygonal in shape, these stress concentrations are considerably reduced. It would appear from the model tests that the circular roadway section is much less favourable than has been assumed previously, since, in this case, heavy compressive stresses are concentrated on the sides of the section in addition to the tensile stresses at the roof and floor. The most favourable cross-section lies between the elliptical and circular section, as shown in Fig. 145B. Supports in this form, however, are difficult to make and it has the disadvantage that the useful width of the roadway is relatively small. Full advantage of these experimental results has not yet been taken.

The trapezoidal or circular cross-section can be used in all cases where only small or moderate pressure conditions are anticipated to exist. Where heavy pressure conditions are expected, preference should be given to the arched section and, as far as it is possible to introduce it, to the 'pear drop' section, elliptical and pentagonal forms of support.

(*c*) *Dynamic stress.* Apart from the stresses produced by the static pressure due to the cover load, the main and auxiliary roadways, as well as other working areas, are subjected to the effects of the extraction of the seams above or below them. The resulting

movements affect not only the surface, but also the intervening strata beds. Thus, roadways lying within the actual zone are also affected, and will be lowered and exposed to horizontal and vertical compressive and tensile stresses. Since this stress redistribution is caused by strata movements, it is known as dynamic stress.

The full effect of the dynamic stress is not confined to the road-

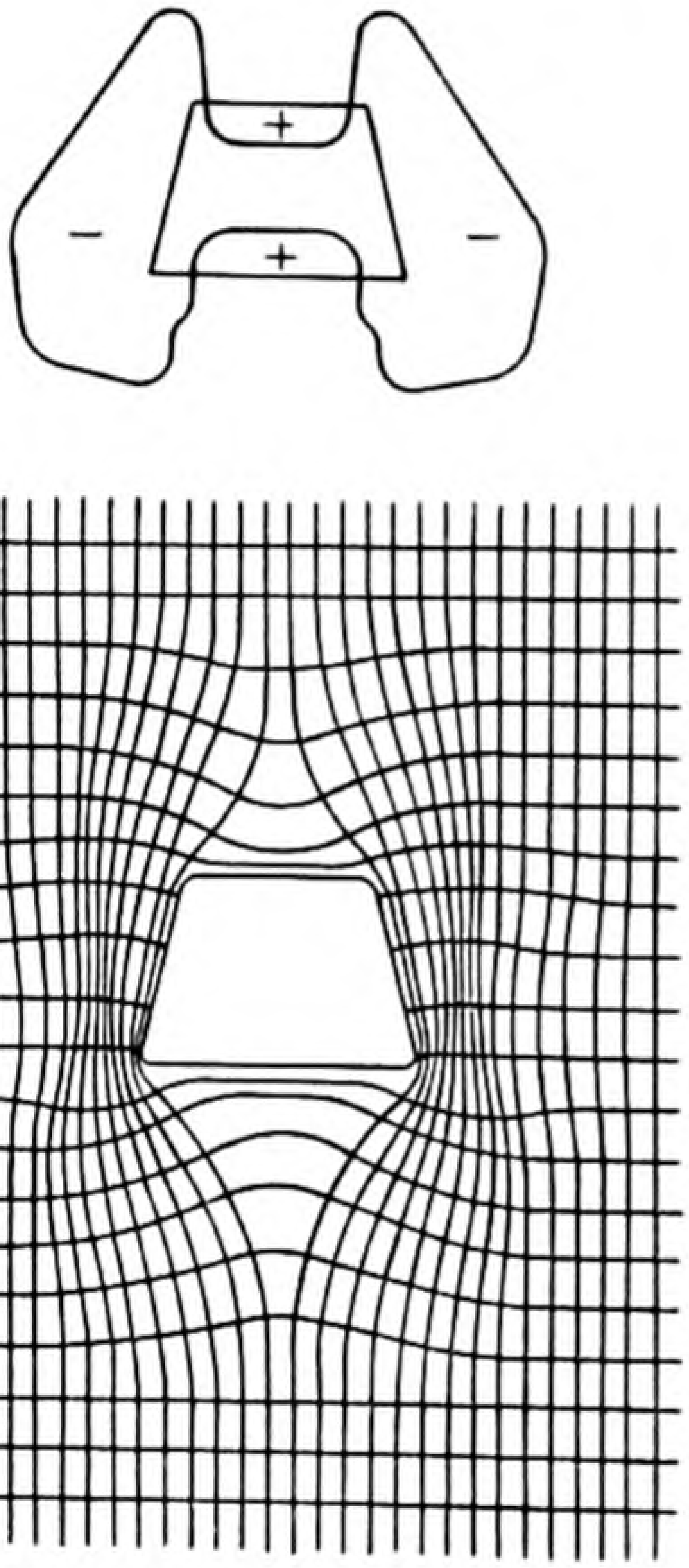


FIG. 145A

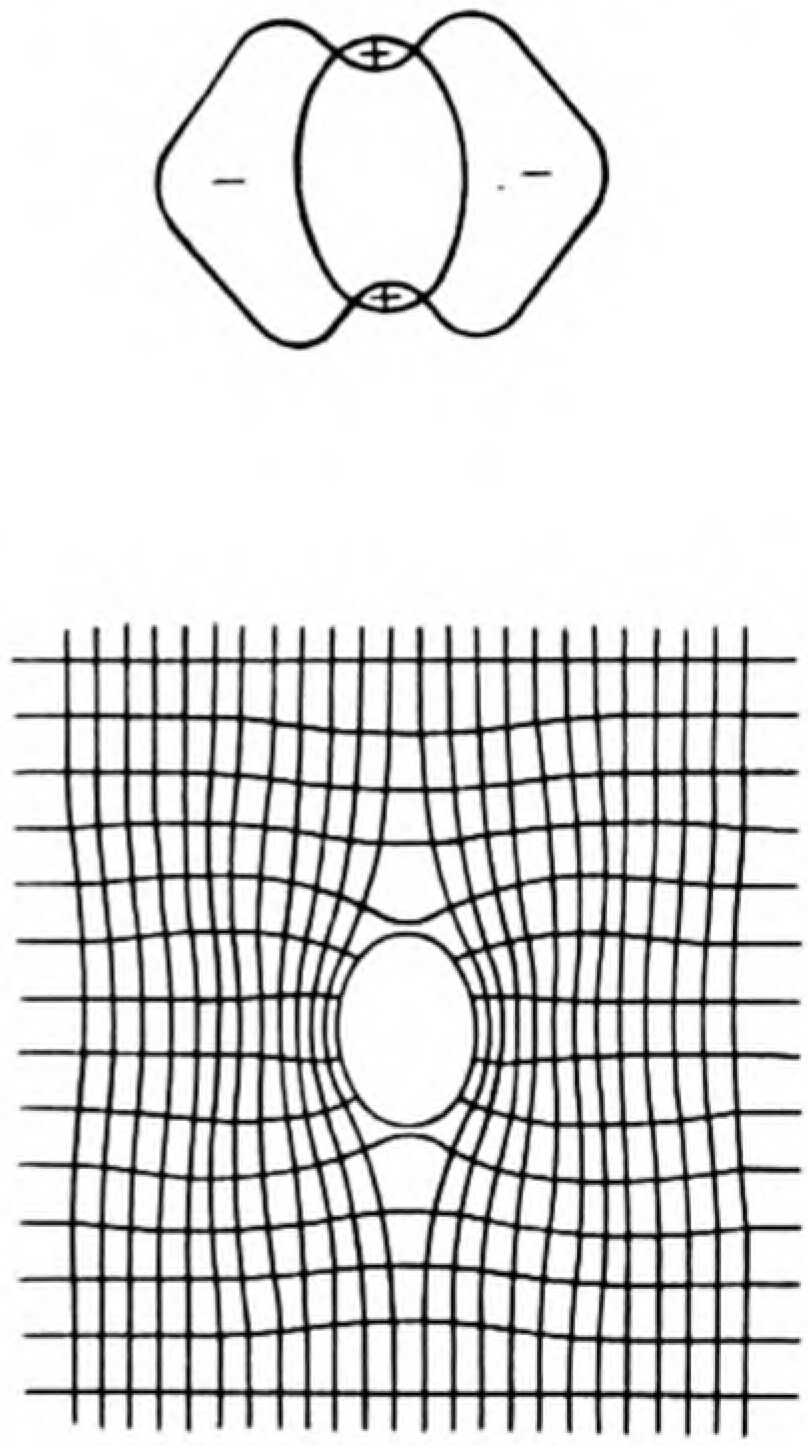


FIG. 145B

ways above any underlying excavation or working, since the stress has also lateral and vertical components acting sideways and downwards. This is illustrated clearly by the fact that zones of increased stress are formed around the working area, and that the changes in the stress distribution within these zones and the working area proceed in a downward direction. It has already been explained that an increase of stress in the direction of the floor of the seam is possible from the effect of pillars remaining within a mined area due to (*a*)

difficult mining conditions, (*b*) safety pillars or (*c*) for the protection of main roadways (Chapter 2, Part 3, Section 2).

The effects of the dynamic stress on the sides of the working area are clearly seen in gate roads from 30 to 60 yards in advance of the working face. In the downward direction below the working area, these stresses will affect mainly the stone drifts forming the main haulage road, causing uplift in the roadways and increased compressive stress on each side of the area. The smaller the vertical distance between the haulage road and the working seam above, the more pronounced the effects will be. The total effect downward is smaller in extent than in the roof beds, since in this area the movements are greater.

Comparing the resulting effects of the static and dynamic stresses upon a net of roadways below a working area, there is no doubt that the latter are more severe and that roadways which are only under the influence of increased static loading can have a much longer life and require less maintenance. These factors will be increased immediately a dynamic stress produced by working near the roadways is introduced. The static load always remains constant, while the dynamic load is fluctuating and liable to periodic changes to a maximum and minimum effect depending upon the advance and retreat of the working from the affected area. The dynamic effect is reproduced again as soon as further working is commenced in a second or third seam within the vicinity of the same area.

Section 2. Types of Support in Stone Drifts

Stone drift supports vary both in the material and the form of support introduced. The supports can be constructed entirely of timber or steel or a combination of both; roadway lining materials such as ordinary brick and concrete are used sometimes and reinforced concrete or precast concrete blocks have been used.

The form or shape of the support can be rectangular, trapezoidal, arched, circular, polygonal or ellipsoidal.

The supports can be designed to have a certain flexibility of shape by yielding devices or loose-jointing or a combination of these methods. Yield can be incorporated in the support by setting it on a yielding material or including the measures to provide the yield in the support itself. Timber is the commonest yielding material and is

used as crushing timber, and for tapered props or for chocks. The arched or circular steel support can be designed to give a certain degree of yield so that the segments can move relative to each other, as in the Toussaint-Heintzmann support. Where the legs of the support penetrate the floor, indirect yield results. Steel supports can also be designed to bend under the roof or side pressure, since, if deformation is kept within certain limits, the support will still provide a resistance to pressure.

Loose-jointing can be defined as existing when at least three sections at four joints, including the ends of the side segments, are designed to move relative to each other. A typical loose-jointed support is the polygonal form. The doorstead form is not loose-jointed, since the bearing areas between the cap and the props do not allow movement. Forms of support in which crusher timber pads are used have little loose-jointing effect, as, for instance, the concrete segment support.

The normal steel arch or circular ring is a rigid support. Similarly, a doorstead support of non-yielding wooden or steel members is considered to be a rigid type of support. Where concrete is used without including crusher pads in the lining, the support is also of the rigid type.

The advantage of a type of support which incorporates some flexibility of form is that it can accommodate the variation in the strata pressure to a certain degree. Such a property is especially important in the case of a roadway subjected to dynamic strata pressure. In this case the pressure is alternating, causing movements and changes in the cross-section of the roadway. The loose-jointed support is able to accommodate itself to such changes, and the flexibility need be only sufficient to enable the support to absorb these dynamic changes for as long as they are present. Where the static pressure of the strata only has to be considered, a rigid type of support is preferable. Where dynamic pressure can be anticipated, loose-jointing and a certain degree of yield is an advantage. A large degree of yield is undesirable, since the cross-sectional area of the roadway would be affected, with consequent difficulties both in haulage and ventilation roadways. The loose-jointed support may have the disadvantage that the destruction of the support sometimes starts at the joints. In stone drifts, rigid or loose-jointed supports or supports with a minimum of yield are to be recommended.

The types of support can be discussed under the separate headings:

(*a*) *The support of stone drifts with timber.* The quality of timber used for underground supports is important. The timber should be dry, tough, solid and durable. Acacia is better than oak, while copper beech and conifer types follow. Props and wood chocks require to be strong in compression, and caps must be able to withstand bending. The direction of the fibres should be parallel to the applied load when the timber is used in compression. The narrower the annular rings are in the timber, the greater is the bending strength. Curved and knotted timber should be avoided, if possible, as the strength is reduced considerably by their presence. Timber used in roadways in which the air is humid should always be treated to prevent attack by fungi.

The doorstead support is the usual form in which timber alone is used for the support of main roadways. The support consists of a cap and two posts, which can be set either normally or with scarfing. Since the posts are weakened by the scarfing, the thick end of the post is put at the top and, in consequence, the weaker point is at the bottom where yield is required. Generally the ends are untapered but bluntly chipped. Crushing pads are required between the cap and the roof above the posts and at the upper ends of the posts between the posts and the roadway side. The pads provide sufficient space for lagging if this is required. In order to avoid movement of the posts due to side pressure, they should be set slightly into the floor.

Since main roads are usually of large cross-section, over 100 square feet, the disadvantages of the wooden doorstead type of support is obvious. The variation in the strength of the timber and the load placed on a long wooden cap, which frequently fails at the centre, have discouraged its use. This type of support is justified only where small pressures are to be withstood and in smaller roadways, such as cross-cuts in sub-levels. Where timber is cheap in comparison with other forms of support, the method may be favoured. An alternative which is more widely adopted is the doorstead support incorporating a steel cap, which often can be of old rail section. In order to use timber and steel together in the same support, it is advisable to incorporate a special joint. The joint prevents the cap from being pushed off the timber posts and protects the softer

timber against damage at the bearing surface between the steel and the timber. Fig. 145A illustrates such a device which is a 'Z'-shaped steel plate called a 'cap angle'. The web of the steel section fits into the slot and the bending gives a certain degree of yield.

(*b*) *Steel supports. Properties of the steel.* The mechanical properties of the numerous kinds of steel available depend upon the method of manufacture and the heat treatment involved. Steel which will withstand only a small elongation before fracture is termed brittle, while steel having a considerable elongation before fracture is ductile. In the case of steel for supports, it is desirable to use a ductile steel which will bend and not break under heavy pressure.

Where disused rail sections are being used, due to their low price, it should be remembered that they may have undergone cold-

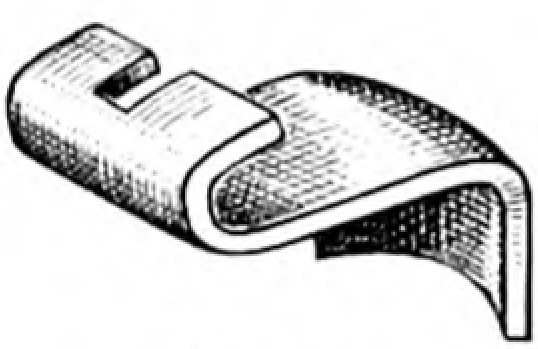


FIG. 146A

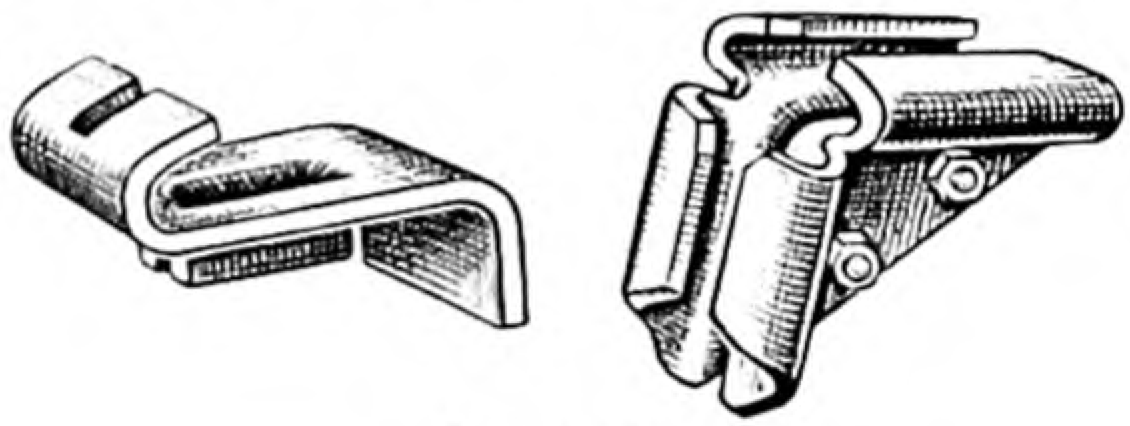


FIG. 146B

working and, if so, their ductility is reduced. They should preferably be heat-treated by annealing at about 820° C. before being put into use, and be carefully examined for hair-cracks.

(*c*) *Types of steel section.* The different steel sections used can be referred to in any structural-steel hand-book.

(*d*) *The steel doorstead support.* In this type of support the cap and post members all consist of steel sections, the disused rail section and the 'I' section joist being the most common. The connection of the props and cap is provided by steel angles fastened by bolts to each member. Since the strength of the support will be affected by the shearing strength of the bolts, this type is suitable only where moderate pressures are to be withstood, or where vertical pressure only is present. Cap-angles or cap-shoes are preferable, and Fig. 146B illustrates two designs. The cap-angle consists of a 'Z'-shaped steel plate, the downward bent sections embracing the web of the prop or post, and the upward bent section fitting into the web of the cap. Another cap-shoe design which has been proved suitable is shown in Fig. 146C.

Although the load capacity of this form of support is consider-

ably greater than the timber form, it is suitable only for larger roadways where moderate pressures are expected. The trapezoidal form of doorstead support is not to be recommended, since the cap-piece and props are always subjected to a bending stress. Where a sudden pressure is exerted on this form of support, an unfavourable distribution of stress will be set up in the framework.

(*e*) *Steel arches.* In this type of support the form of the arch and the splay of the legs can be varied widely. The shape may be semicircular, horseshoe, flat-topped or tapered towards the arch.

Among these shapes, the flat arch is probably the most unfavourable type, since it differs only slightly from the trapezoidal doorstead form. Vertical legs are also unfavourable because they are subjected to a bending movement while in the arch, part of the bending stresses will be converted into a compressive load which the arch can resist more easily.

FIG. 146C

The steel section and weight per foot of the various shapes of arches differs greatly. The common 'I' section joist, wide-flanged girders and special cap sections with reinforced web and flanges are in use. Sections which have been expressly designed for roadway supports are being adopted more widely. The Toussaint-Heintzmann section is typical of the channel form of section, which allows a suitable construction to provide yield. The British 'Twintrof' section is a typical form of section used for both arches and steel props. The heavier sections, due to the difficulty in handling, are used only on roadways which have a long life and where high strata pressures are to be expected. The weight of the steel supports varies between 9 lb. and 35 lb. per foot; weights between 15 lb. and 25 lb. per ft. are most common.

The spacing of the supports is extremely important to ensure maximum life from a set. The distance apart is generally one yard, but it may be increased up to 5 feet in favourable circumstances or reduced to 2 feet where strong pressure is to be resisted. The individ-

ual arches should be strutted methodically to avoid distortion due to stresses in the direction of the roadway axis. The arches should be given sufficient contact with the surrounding roadway sides, either by direct contact through lagging, or strong packing, otherwise the unsupported arch section may buckle at points of minimum resistance where the excavation is wide.

Each steel arch generally consists of two or more segments. The connection can be a weak area unless a strong connecting piece

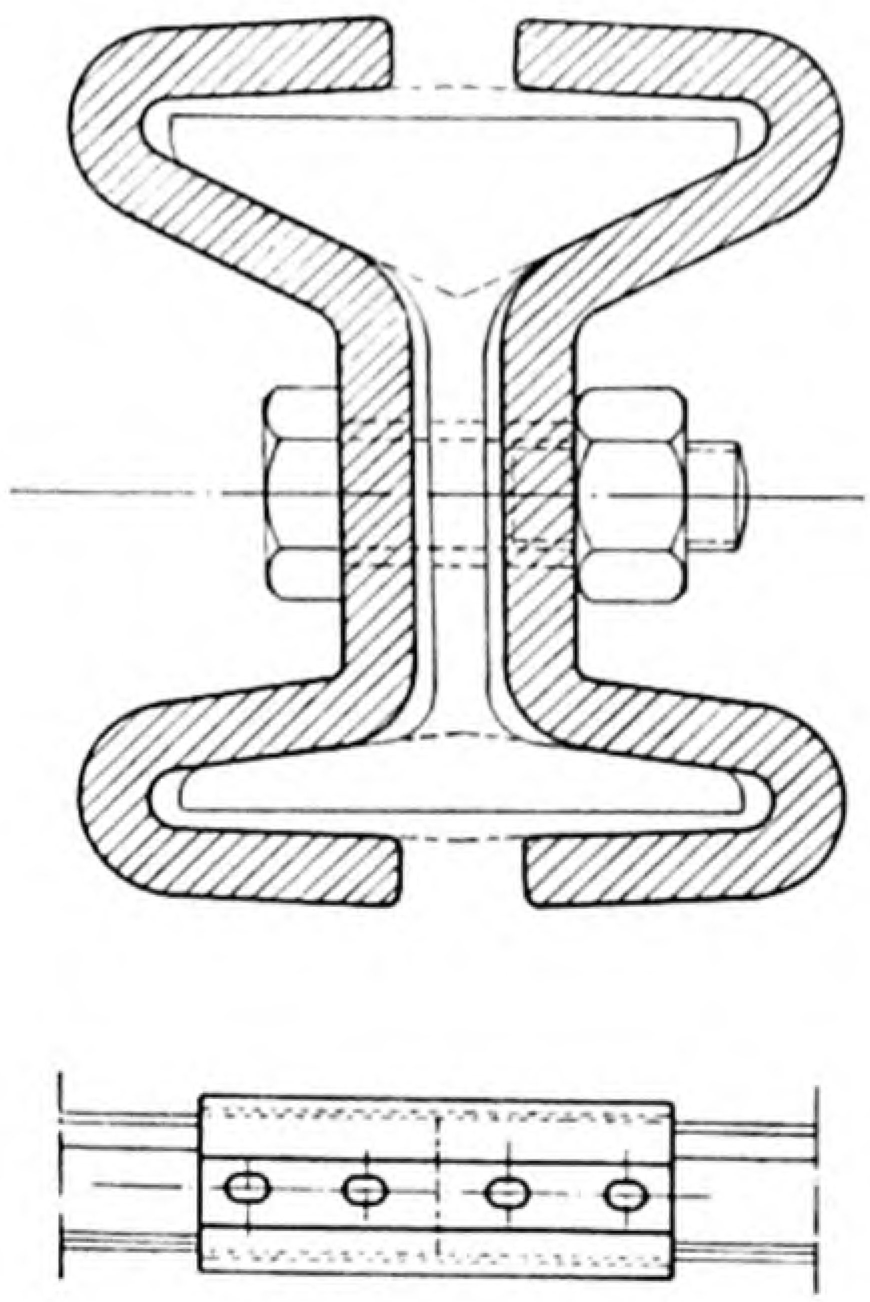


Fig. 147

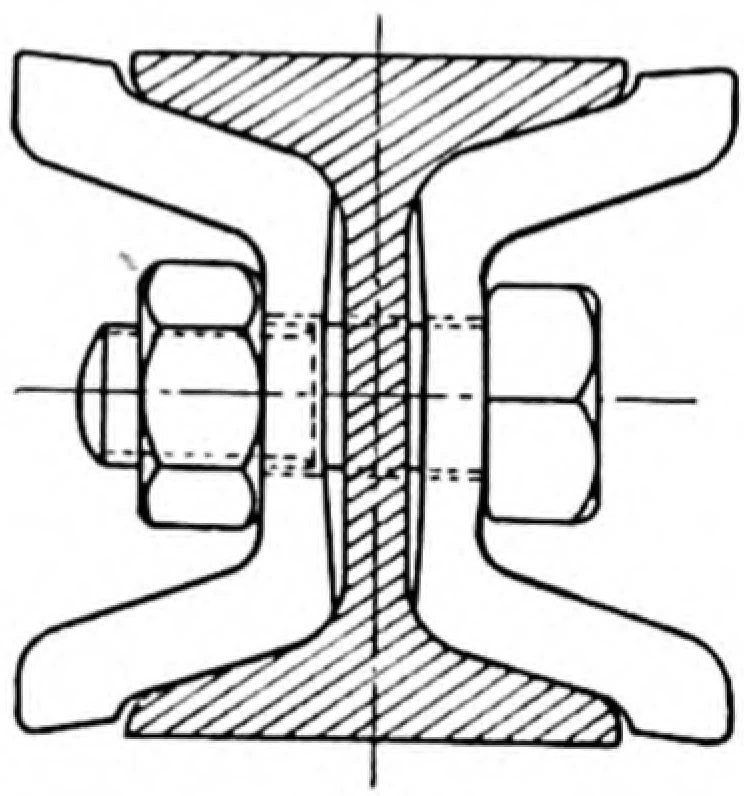


Fig. 148

is used. The cheapest and least durable form is a pair of flat fish-plates, bolted to the arch. The channel form of fish-plate has more resistance. The best form of plate is one shaped to fit the girder section and which can assist in taking part in the resistance to the applied load. Figs. 147 and 148 illustrate this form of clamp-plate. With these plates, the ends of the segments are in contact, while with another form of clamp-plate, with which a gap is left between the segments, a certain degree of yield is obtained, since the segments can be moved towards each other over part of the plate length. Recently, another form of joint has been successfully introduced, in which the 'joint-nut' shown in Fig. 149 is used.

The Toussaint-Heintzmann arch, already referred to, consists of three segments, which, due to the channel section used, can be moved against each other. The most recent design consists of equal rolled sections, as shown in Fig. 150, the form of which cuts out any possibility of seizing of the sections within each other during assembly, the connection between segments being made with a clamp connecting piece as shown. The use of Toussaint-Heintzmann supports in a main roadway is shown in Fig. 151.

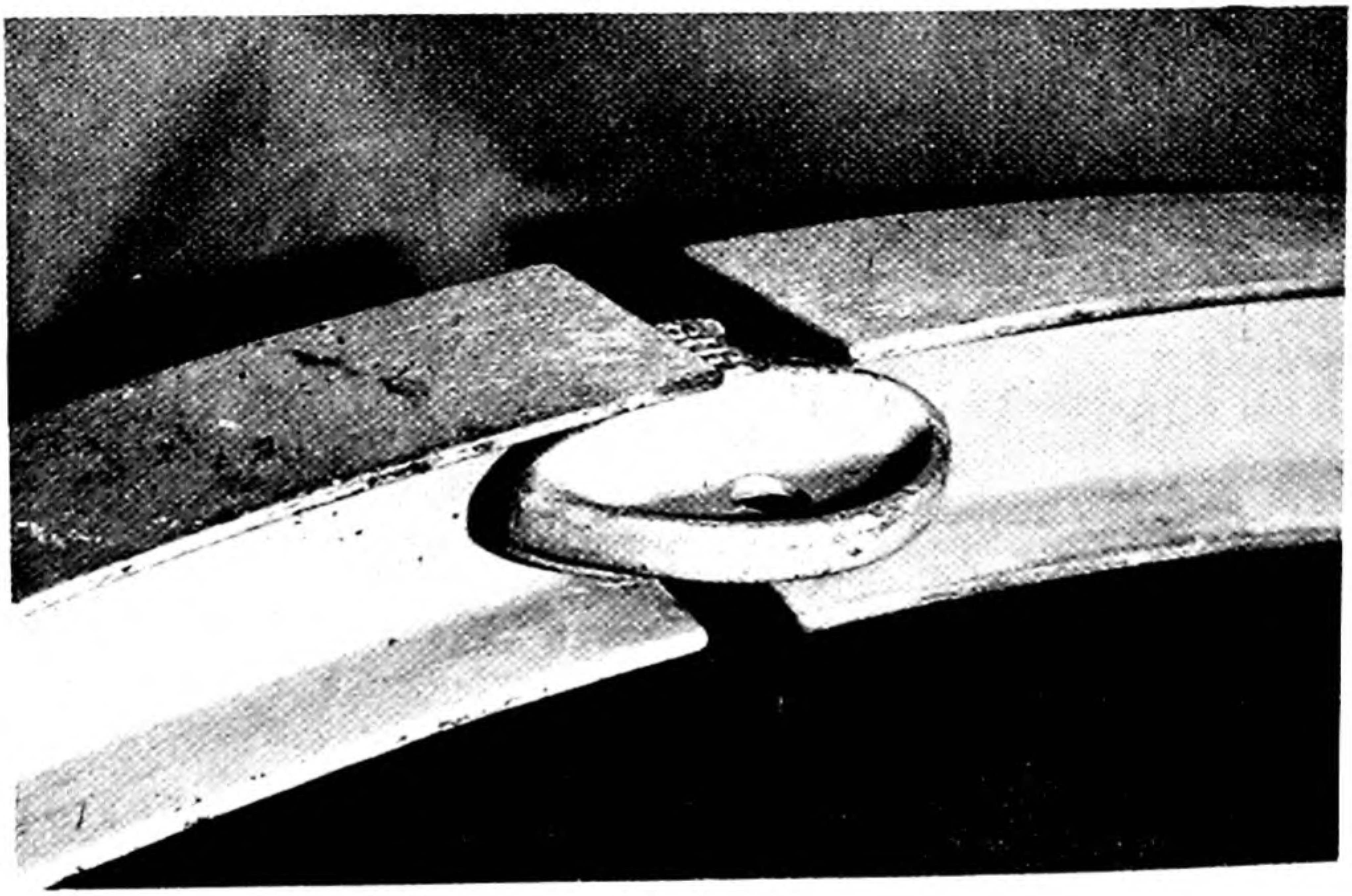


FIG. 149

(*f*) *The steel ring supports*. Whereas the arch support is not continuous but open at the floor, the ring support is closed. The ring is therefore more resistant and is more applicable to places where the effects of creep of the floor must be prevented. Since the material cost is higher and the system requires a greater amount of stone work, it is more expensive than the arch form of support.

The same steel sections are available for this type of support and, as it is more useful where high pressures are to be resisted, heavier steel sections are being used. With ordinary sections, three segments are used, *vide* Fig. 152, while there are four with Toussaint-Heintzmann sections. Thus these supports can be made rigid or

yielding according to their design. Where yielding supports are required, the Toussaint-Heintzmann section is recommended, since, with heavy sections, ordinary flange connections cannot provide sufficient resistance.

(g) *The polygonal support.* This form of support is normally open at the base and built up from four segments. Two curved steel segments form a 'vault', while the other two bear this arch and can be of either steel or timber.

The polygonal support has another advantage apart from its loose-

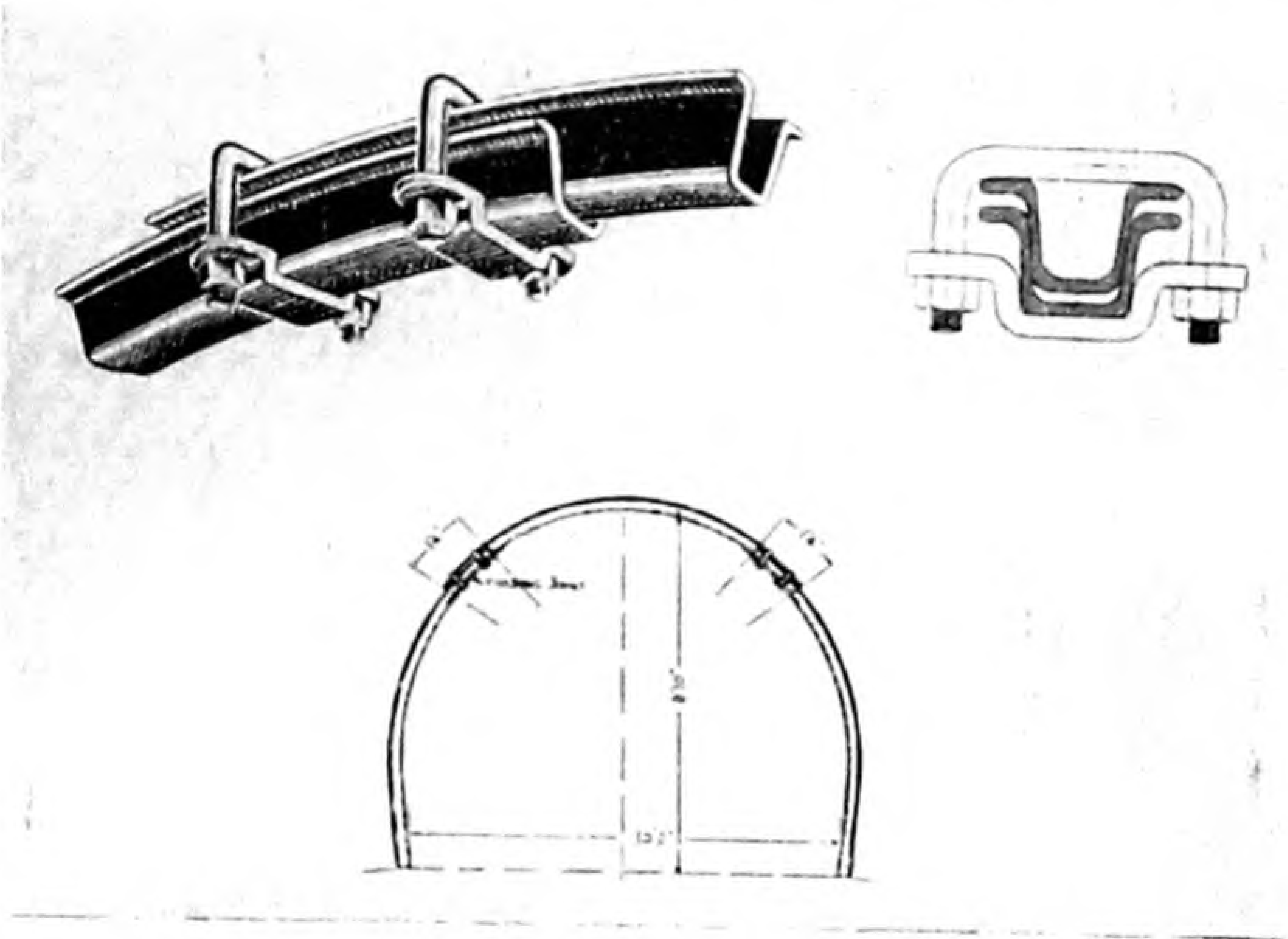


FIG. 150

jointing. Since the joints, which must press closely against the rock sides, take up the strata pressure first, the load is transferred to each section and given a greater pressure against the rock face along each segment. Thus, a major part of the strata pressure is taken up along the longitudinal axis of the segments, putting them in compression, against which the material is most resistant. If the arch sections are also loaded laterally, they must also withstand bending stresses. Due to the curved form of the segments, bending stresses are transformed into tensile and compressive stresses in the members, but compressive loads only are transmitted to the joints. This form of polygonal support became possible only after a satisfactory knuckle-joint between segments was designed. The Moll joint consists of a thick round 'run-

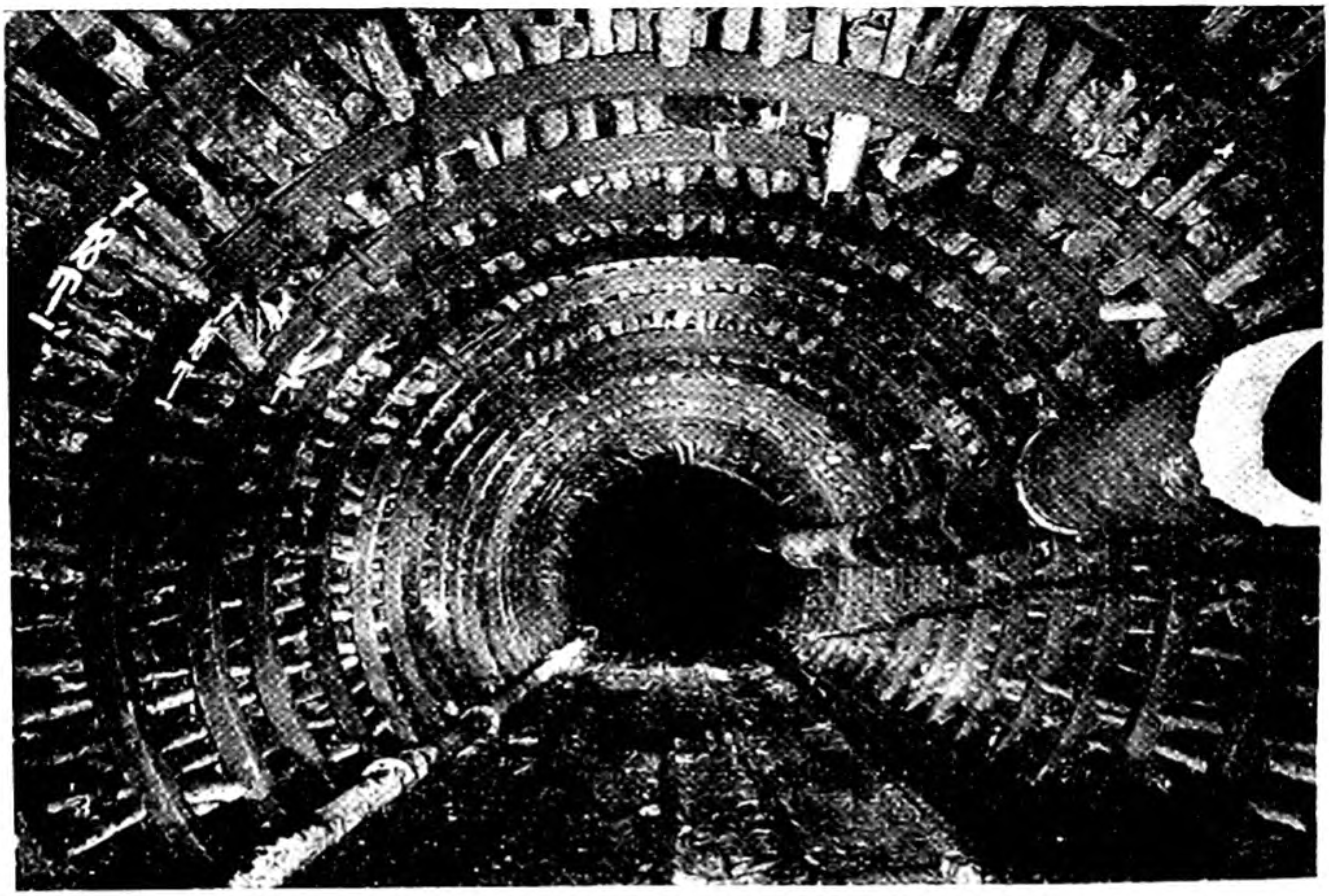


FIG. 151

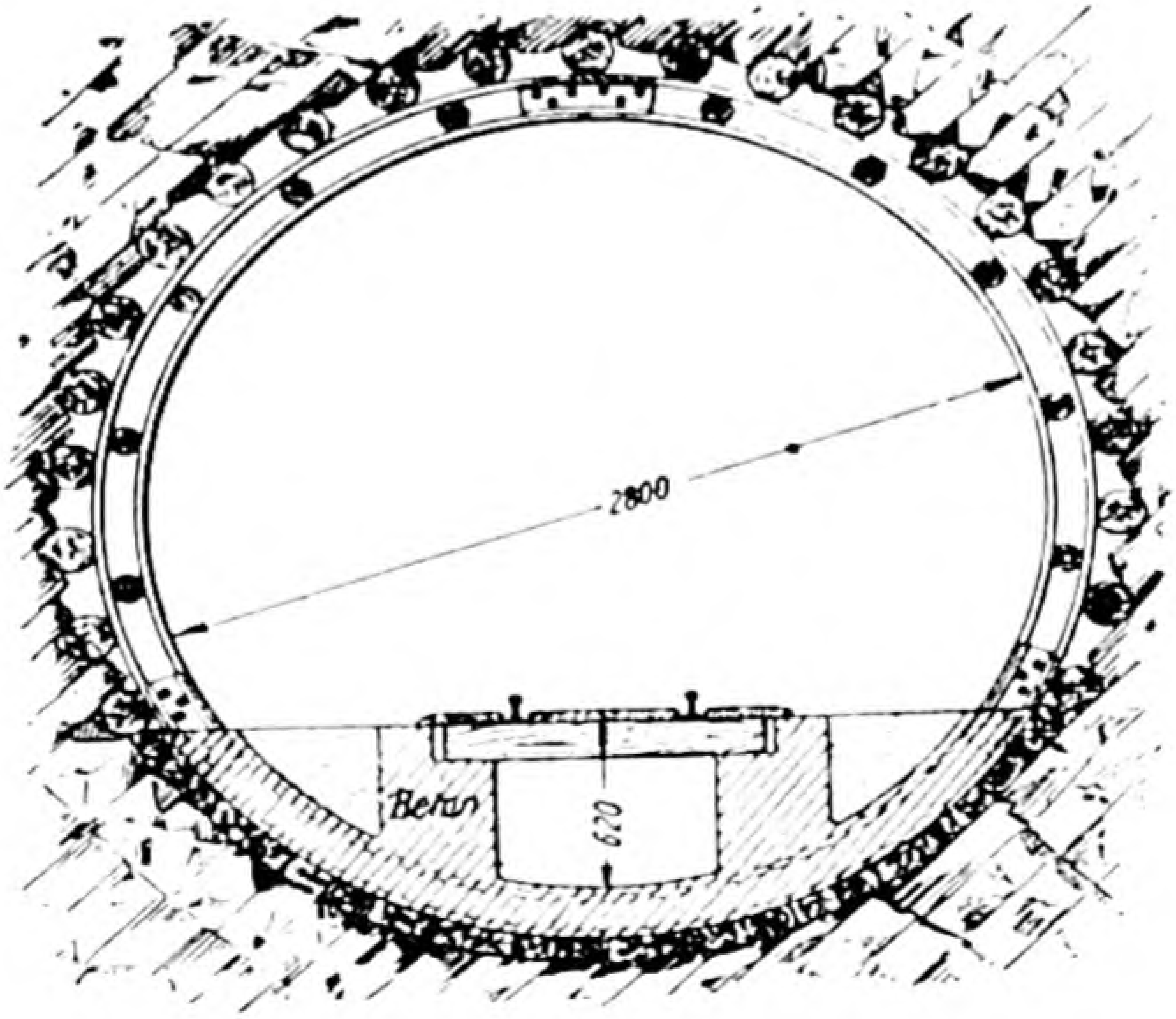


FIG. 152

ner' timber, about 5 feet long, set along the axis of the roadway. This runner is used for two sets and is continuous along the roadway. One runner is set in the centre at the roof and two more along the sides. The essential part of the connection between the steel segments and the round timber are two curved plates or sockets, fastened either by bolts to the ends of the steel segments or welded on. The sockets can be formed integral with the segment. The runners are best made from solid timber, such as oak, since it must not yield but remain rigid and form the knuckle part of the joint, around which the socket can move. The runners are often

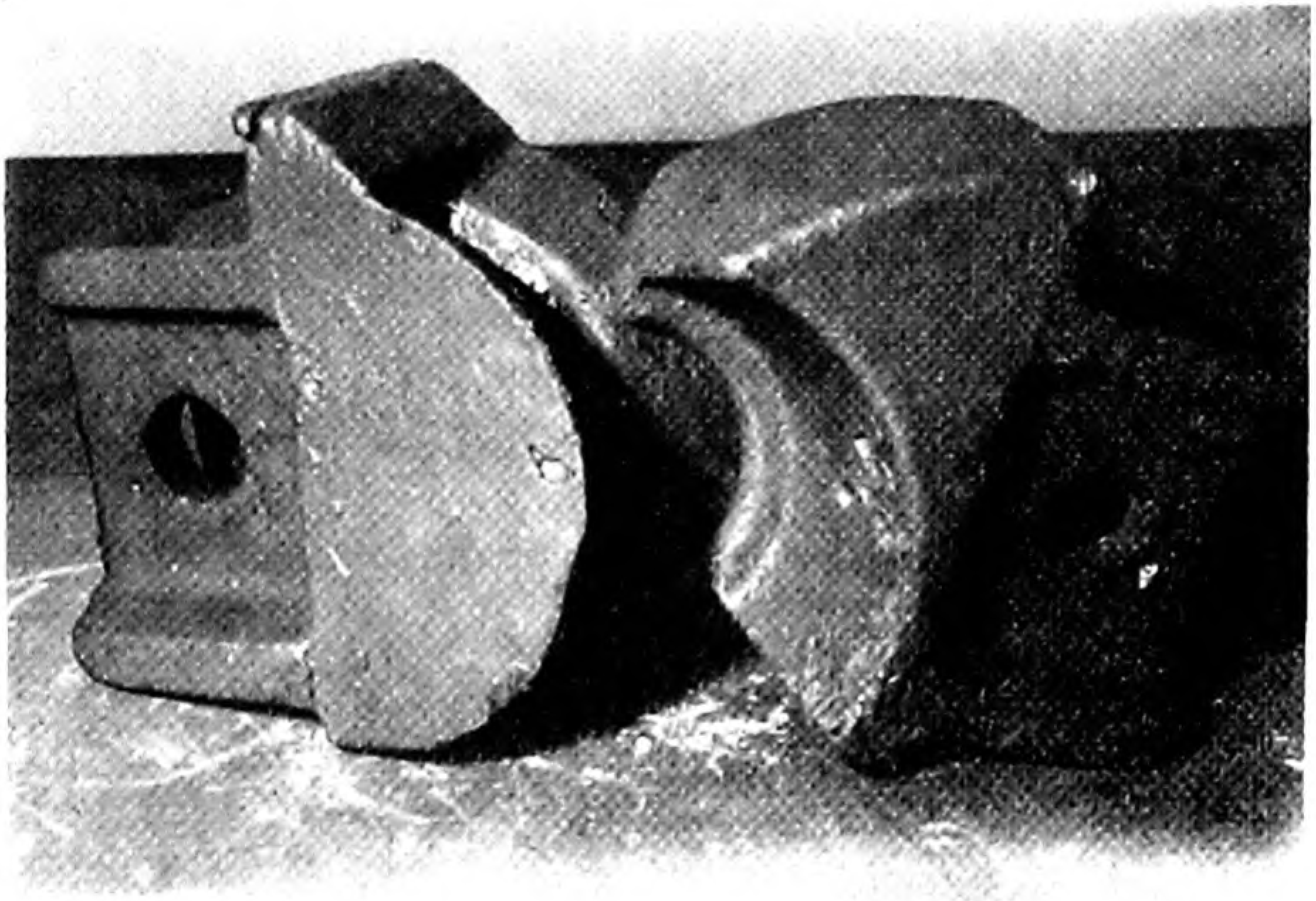


FIG. 153

machined or surrounded by a sheet-steel jacket to prevent their destruction. Special care is required in order to set the supports vertically so that individual joints will come into the correct corresponding positions, vis-à-vis.

Fig. 154 shows a steel polygonal support with timber props, where the connection between the lateral lines of runners and the props has been effected by thick crusher timbers. The timber props can be replaced by rigid steel props.

The illustration in Fig. 155 represents a steel polygonal support with joints of similar construction to those shown in Figs. 159A and B

on pages 232 and 233. Instead of sockets and timber runners, solid steel chill-joints are more and more used. They are mostly swage forgings, fastened to the steel segments generally by bolts, some-

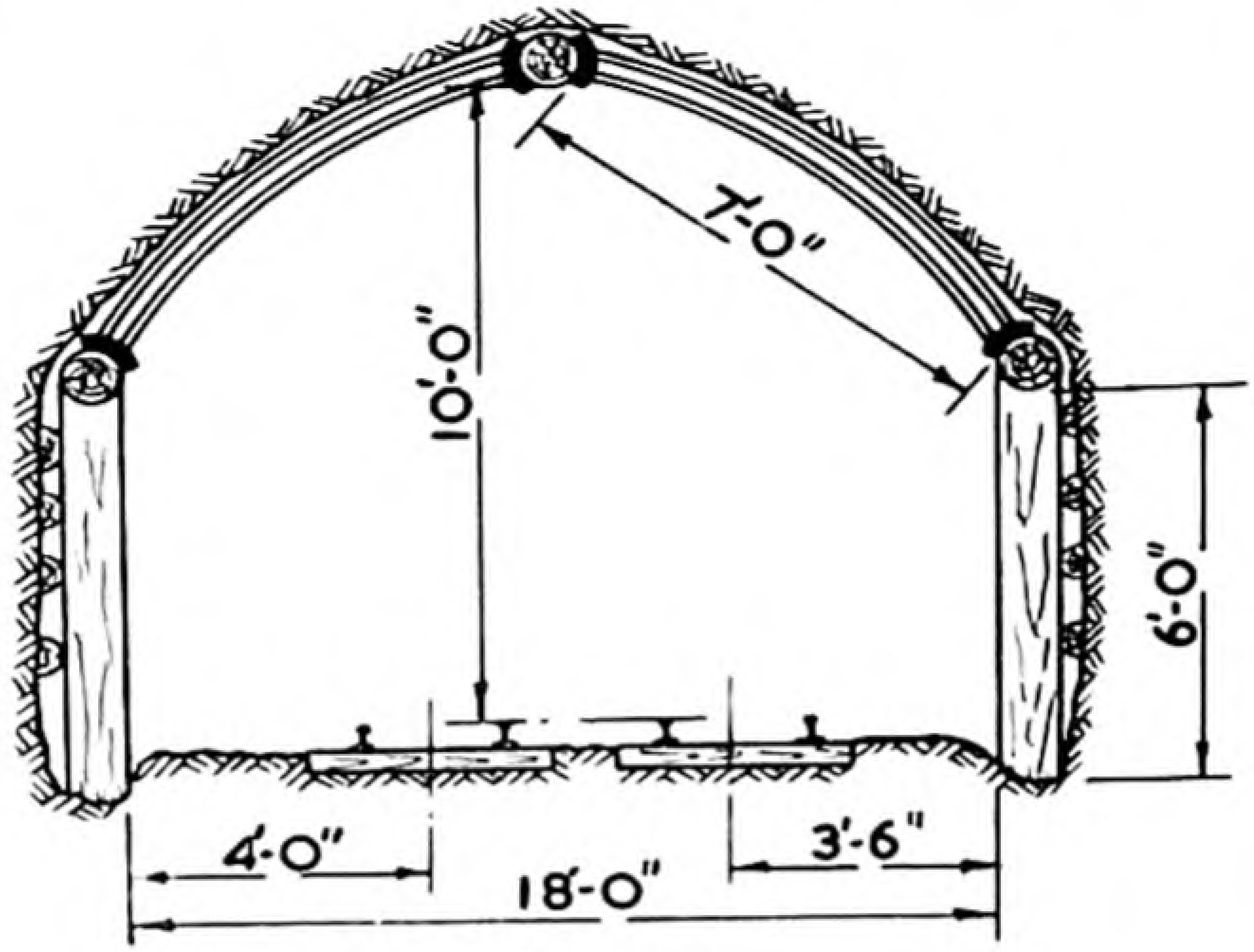


Fig. 154

times by welding. One of the numerous steel chill-joints, used in the Ruhr, is shown in Fig. 153.

(*h*) *Strutting the support sets.* The supports are exposed not only

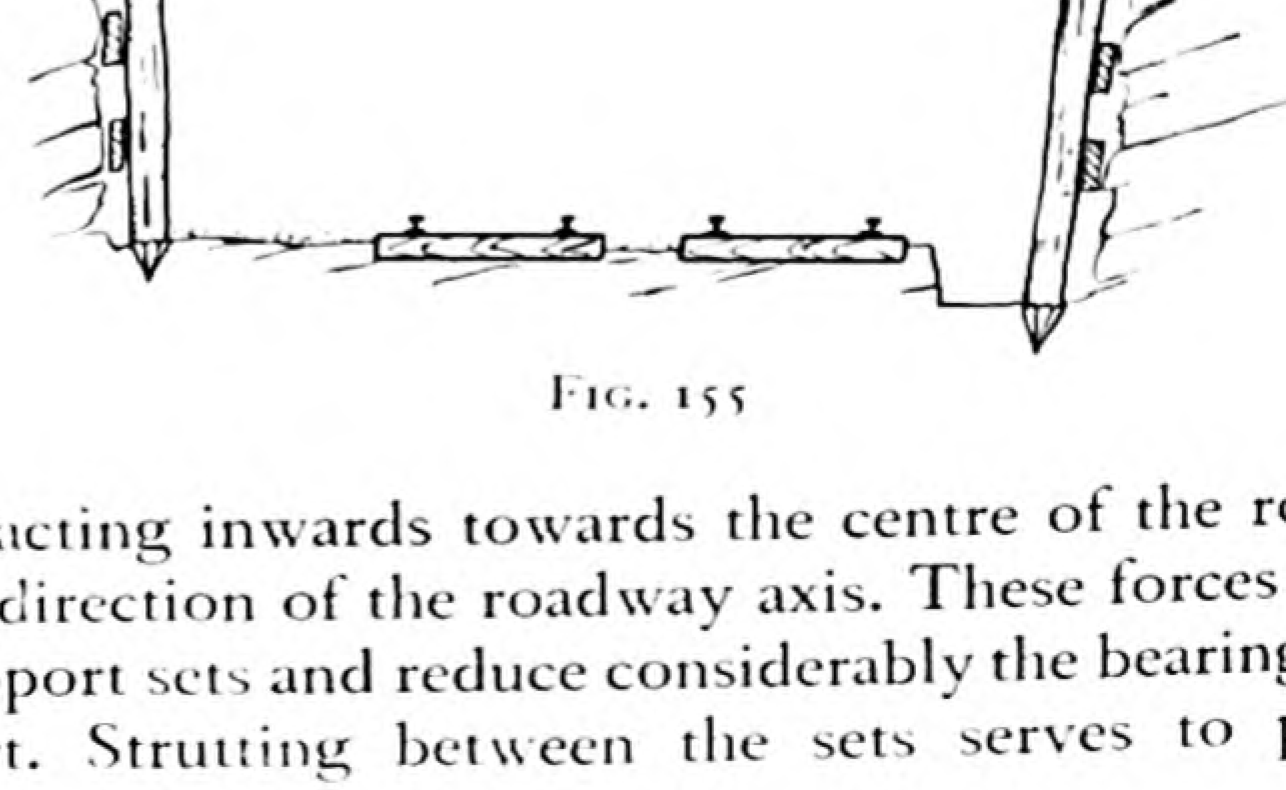


Fig. 155

to stresses acting inwards towards the centre of the roadway, but also in the direction of the roadway axis. These forces tend to distort the support sets and reduce considerably the bearing strength of the support. Strutting between the sets serves to prevent any

individual distortion and makes the support continuous along the roadway. In the case of steel sections, long bolts can be used. It is particularly important to insert the struts in a continuous line between sets so that the sets are not exposed to bending and shearing, as shown in Fig. 156. The distance apart and the number of struts inserted depend upon the nature of the pressures expected in the

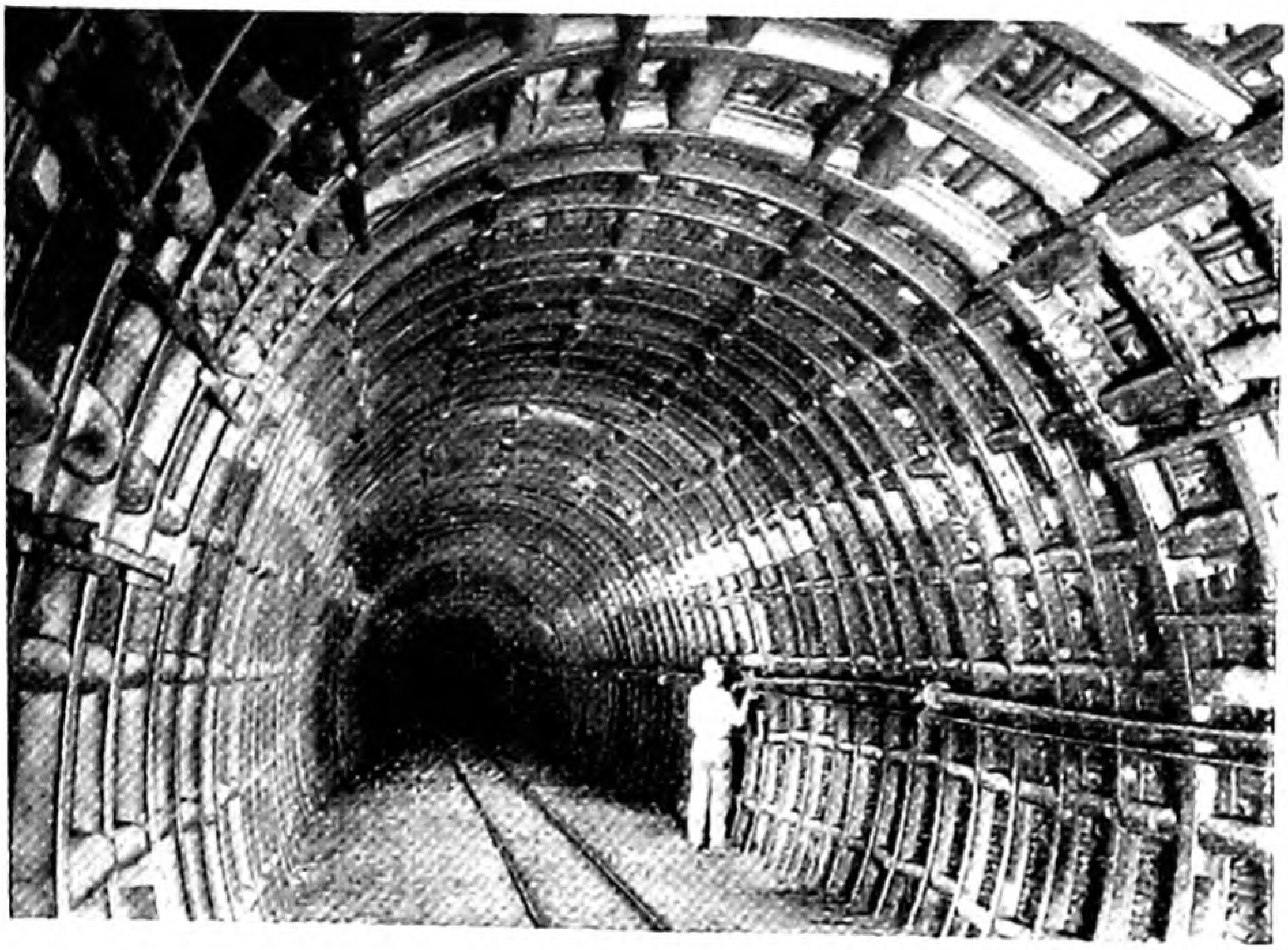


FIG. 156

direction of the roadway axis. It is generally sufficient to insert them at distances of from 2 to 4 feet apart. In general, round timber is used for struts. In the case of all steel supports, timber may be replaced by flat steel bars which clamp round the support section with thin ends bent to prevent reciprocal movement of the sets, *vide* Fig. 157. With the Toussaint-Heintzmann section, round timber should be eliminated, since the lack of flanges does not allow the struts to be kept properly in position.

(*i*) *Lagging the supports.* Since the support sets are only supporting the immediate strata at intervals of from 2 to 5 feet and are not a closed support, it is usually necessary to insert lagging in the intervening spaces to transfer the pressure acting between the frames on

to the supports. The lagging can be of either timber or steel. Split or round timber is used normally when timber-lagging is adopted. The length of the lagging should be slightly more than the distance between sets. In the case of split timber, the smooth or round side is placed next to the rock face, depending upon whether coniferous or leaf timber is used. Coniferous timber has a greater resistance to stress against the sliced side, while leaf timber (deciduous) should be placed with the round side against the rock face. Planks have been

FIG. 157

used for lagging, but are more suitable for staple shaft lagging than for roadways.

Experience with sheet steel has shown that a longer life is obtained than with timber, and it has greater resistance. Scrap steel may be used in the form of steel sections, pressed steel tubes or lengths cut from winding rope. The ordinary flat steel or corrugated straps have also been used, as shown in Fig. 157. Wire mesh has proved useful in some cases. The wire is usually 10 to 5 S.W.G. and laced into a wide-meshed net. Generally, the lagging of the supports is combined with careful packing of the excavation between the rock face

and the lagging. This packing is important since loose stone cannot break the lagging, and accumulation of methane behind the supports is prevented. With careful packing, the supports have a close connection with the strata and the possibility of deformation of the supports is reduced or eliminated.

Section 3. Support of Roadway Junctions

At roadway junctions and crossings, a large area of roof is exposed, and where several roadways meet, the strata is stressed more than is usual, so that these positions must be supported carefully.

The doorstead support can be easily arranged to give adequate

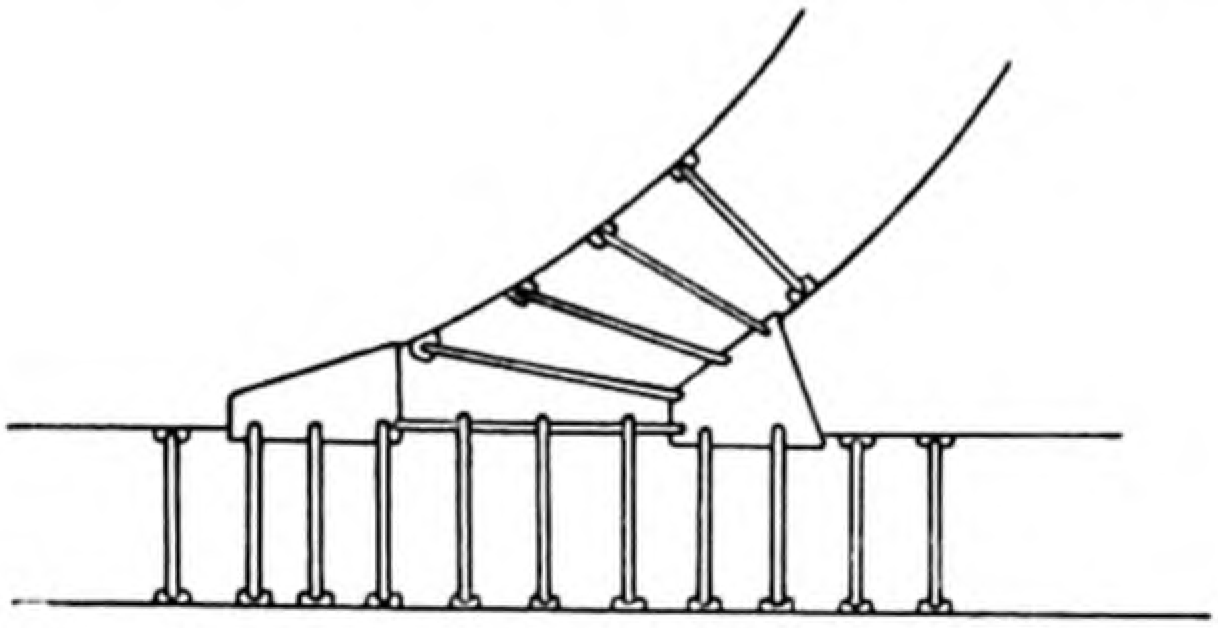


FIG. 158

support to a roadway crossing or junction by the introduction of long cap supports and the use of steel runners, on which one or both ends of the caps can be placed, *vide* Fig. 158. Other forms of support make the solution of this problem more difficult, although the above form of support can be used at roadway junctions even where another form of roadway support is in general use. Brick or concrete walls or filled timber chocks may be used as supports for the steel running girder on which the cross-caps are set. This method is useless where heavy pressures have to be resisted; the disadvantages of a long span are obvious. Brick arches are resistant to static strata pressures, but, on a basis of cost, cannot be recommended unless the roadways have a long life.

Combined with the polygonal type of support, top crowns, either with or without side walls, have proved to be very useful. As shown in Fig. 159A, the quadrangular crown is supported by four double steel stretchers, resting at the upper end on the steel frame of the crown which is timber-filled, and at the lower end on short round

timber set on square timber chocks. The top crown may consist of a welded octagonal frame as in Fig. 159B. The steel segments bearing the crown push through appropriate openings in the top against a hard wood filling behind the crown.

Half-top crowns have been used for single-sided rectangular roadway junctions as shown in Fig. 160, and the top crown leans against the stretcher which has been laid along the roof of the main roadway polygonal support. The support is trapezoidal, consisting of 'U' sections, and is filled with hard wood. Two pairs of diagonal

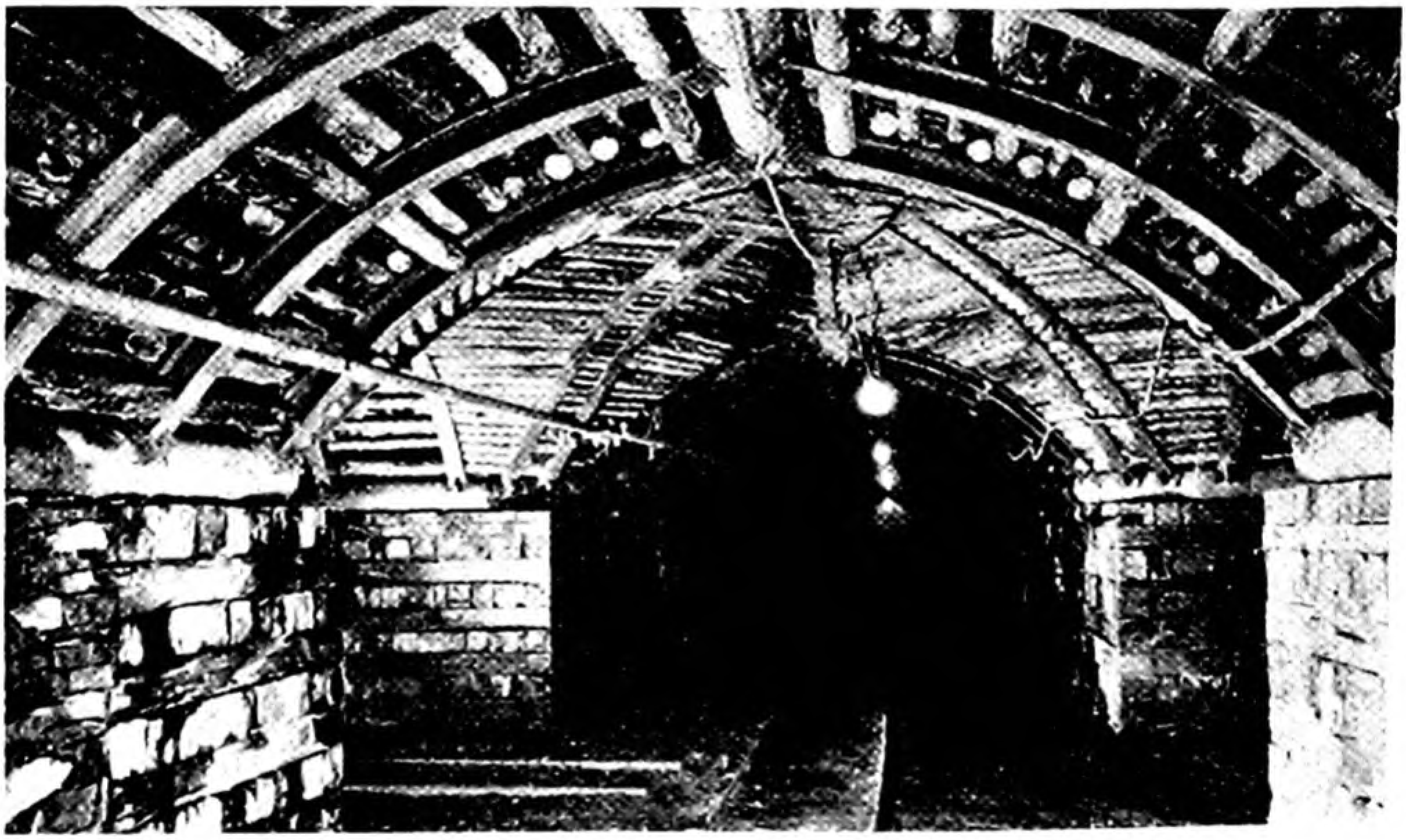


FIG. 159A

polygonal segments are pushing against the 'U' sections, which at the lower end are resting upon short stretchers borne by props or upon a wall. Two more steel sections support the roof between the top crown and the first segments of the normal polygonal support of the roadway branch. Where the roadway junction runs off at an acute angle, triangular frames or sets can be used as in Figs. 161A and B. The enlargement of the roadway in the direction of the roadway branch is being carried out without any increase in height by the introduction of the triangular frame. One top stretcher is resting on each side, against which the segments of the roadway branch are set. The end of the frame is a semicircular arch, against which the segments

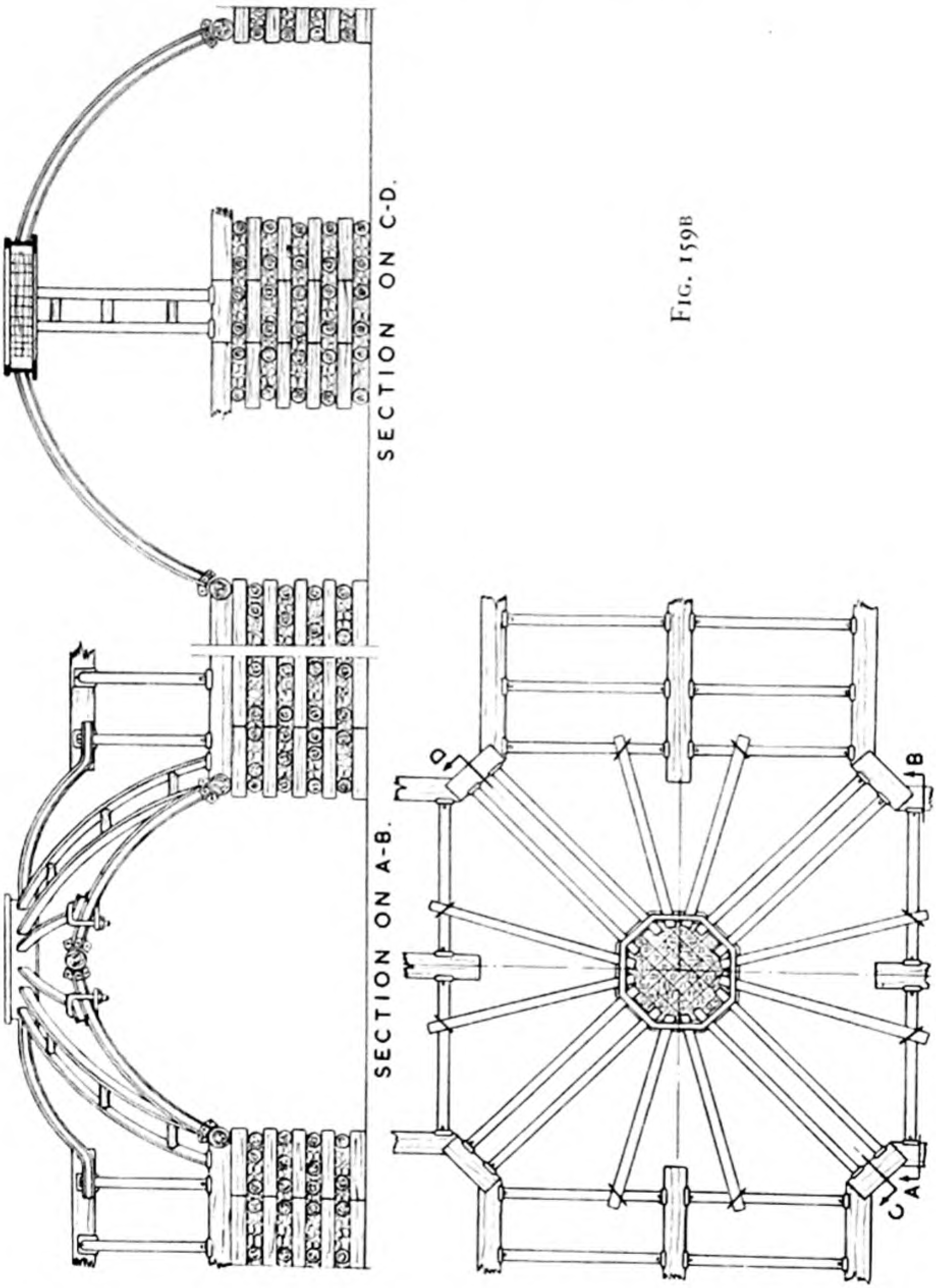


FIG. 159B

are leaning, and the lower ends are resting upon stretchers supported by a side wall. The wider the triangular frame, the larger is its horizontal surface and the more unfavourable is the support to the influence of bending stresses. Branches with a width of from 30 to 35 feet have been supported satisfactorily in this manner.

Section 4. Brick and Concrete Lining

The common brick is widely used for the construction of supporting roadway lining. The standard size of bricks is normally taken as $9 \times 4\frac{1}{2} \times 3$ inches. The thickness of brick walling is specified by the number of bricks measured in the direction of their length.

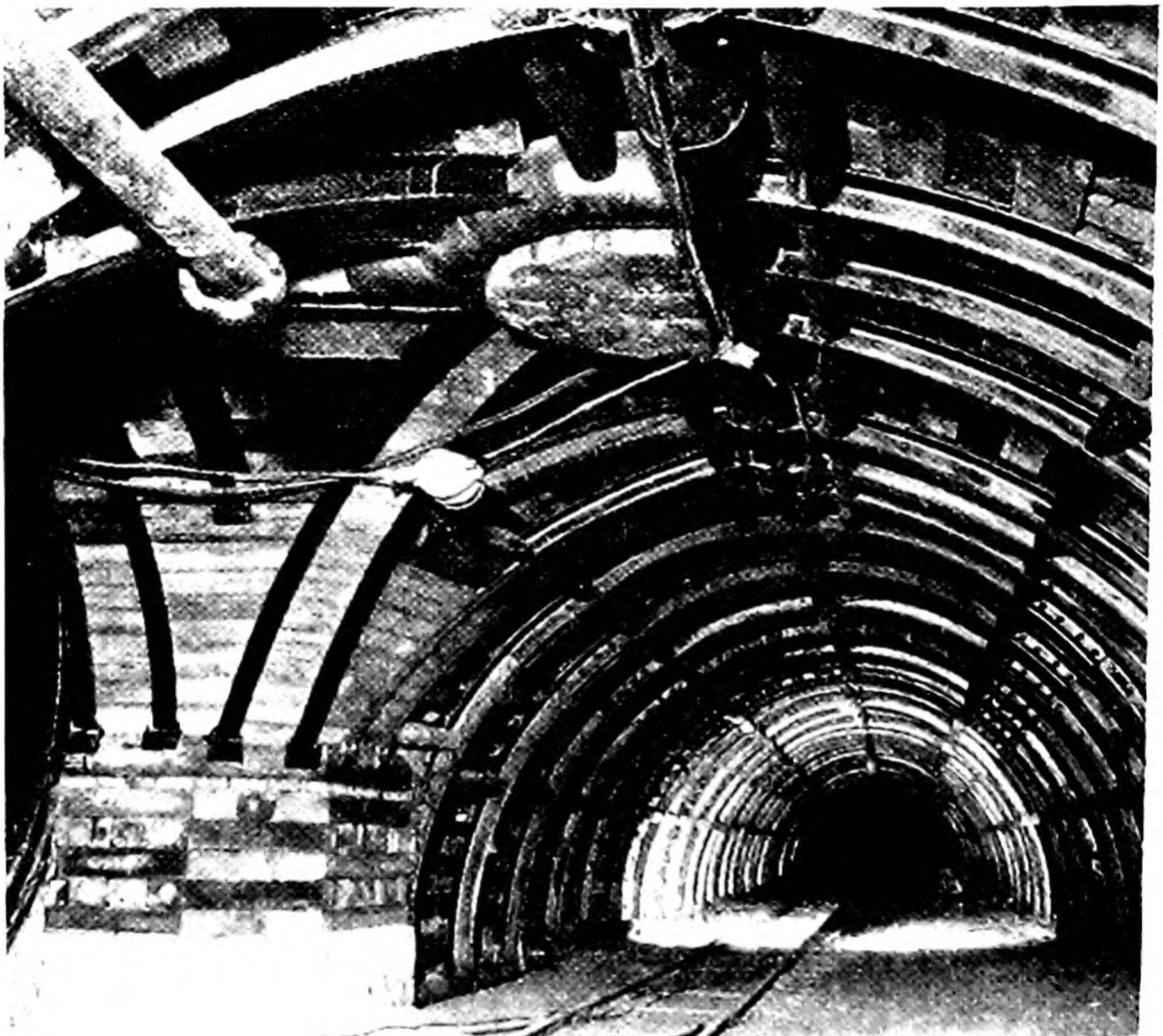


FIG. 160

Thus in a $4\frac{1}{2}$-inch (a half-brick wide) wall, the courses are 'stretcher' courses, and all the bricks are laid facing lengthwise to the outside: in a 9-inch (a one-brick wide) wall, the courses are 'header' courses, the bricks are laid flat with the ends on the outside. Other widths of walling may be termed one and a half bricks wide, etc., in which case the wall is $9 + 4\frac{1}{2}$ inches, i.e. $13\frac{1}{2}$ inches, which is the standard

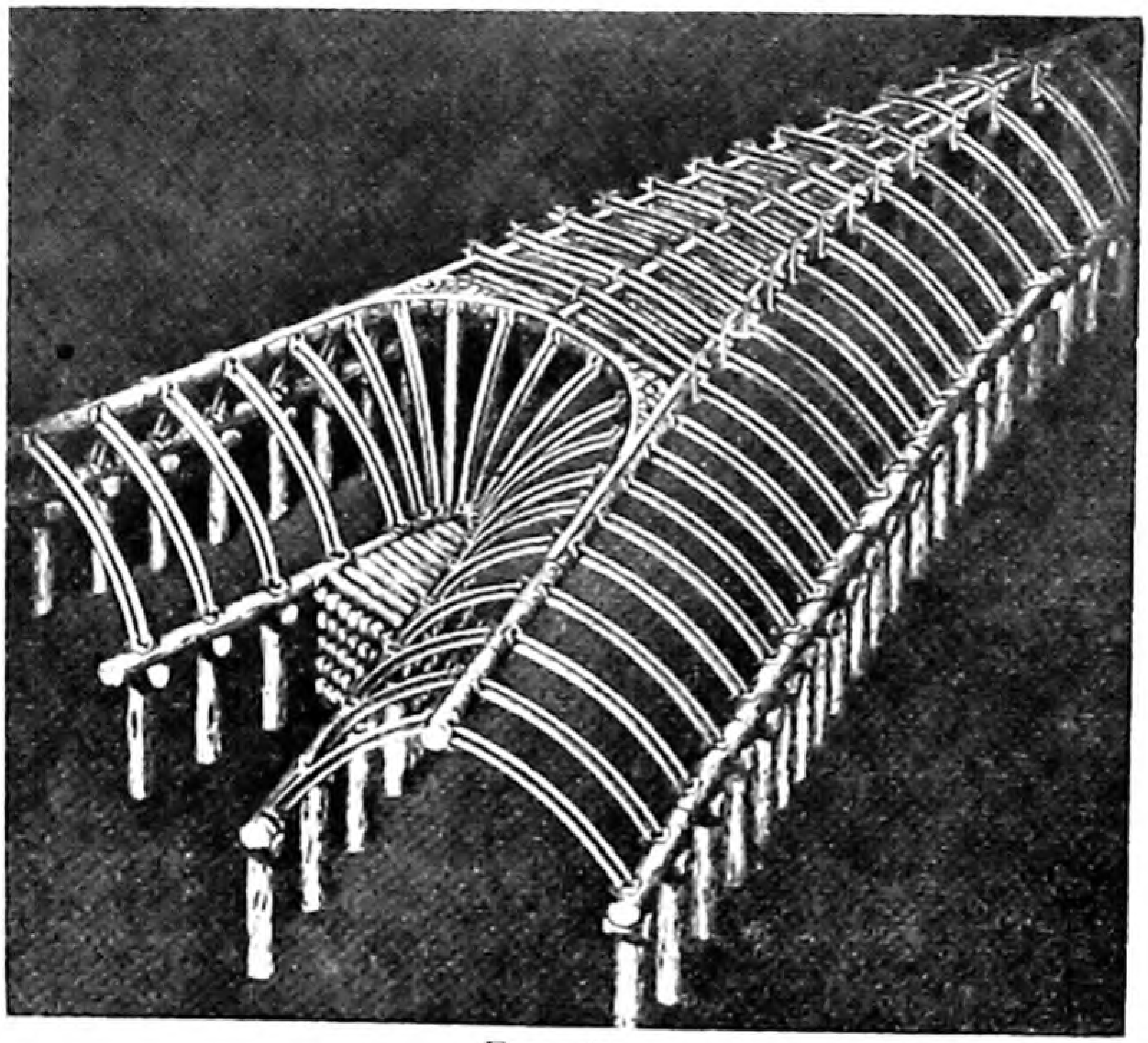


FIG. 161A

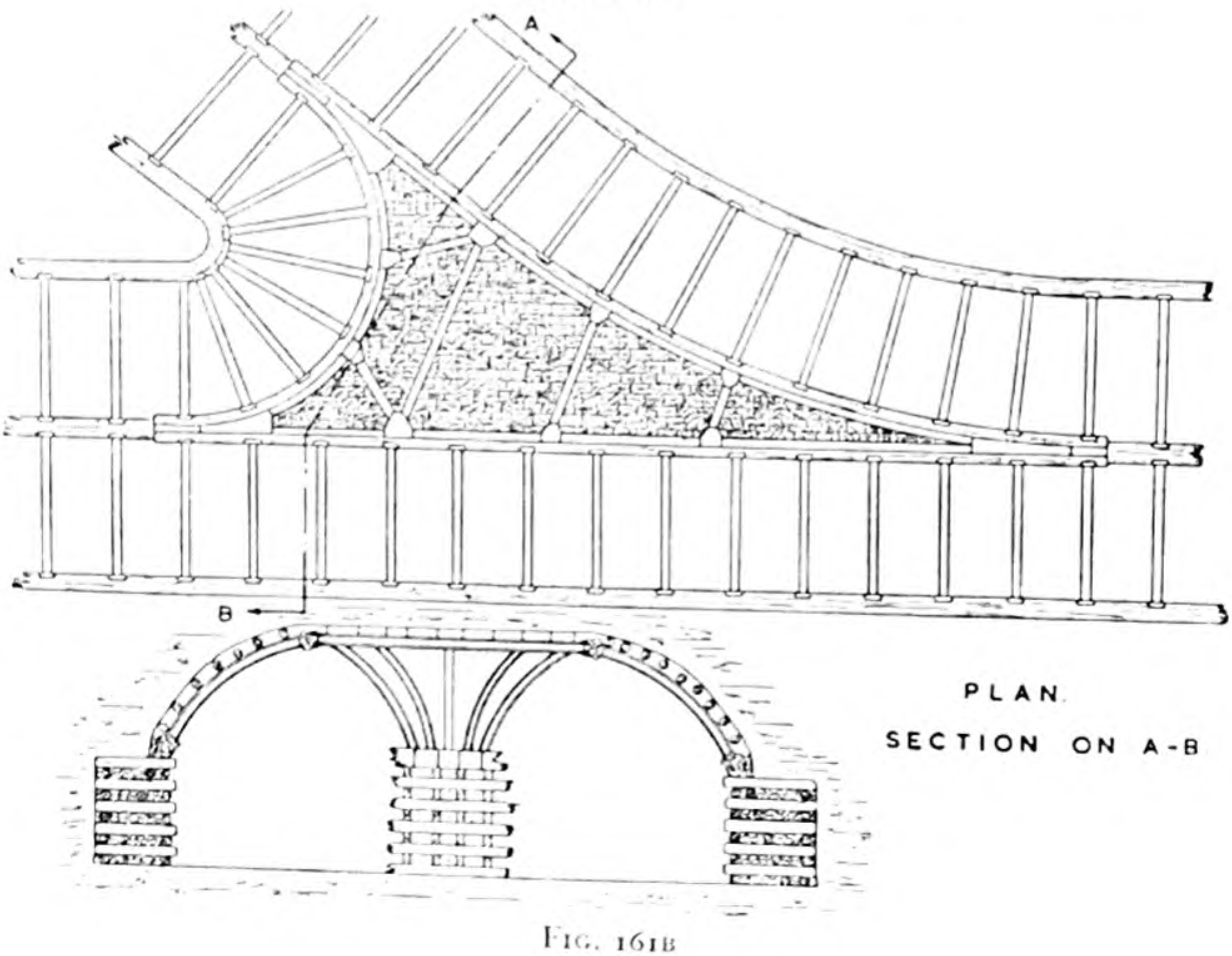


FIG. 161B

thickness. The number of bricks per cubic yard is 384, while 32 bricks laid flat or 48 bricks laid on edge will cover 1 square yard.

In laying the bricks, the vertical joints in header courses must be of such thickness that they will not come over a joint in a stretcher course. For average joints ($\frac{1}{4} - \frac{3}{8}$ inch) 1 cubic yard of common brickwork requires approximately 0·3 cubic yard of mortar, or about 0·8 cubic yard per 1,000 bricks.

The mortars used are generally as follows:

(1) *Normal mortar*, which needs air contact for setting and is composed of lime and sand (under suitable conditions ashes, coke, crushed bricks or crushed stone may replace some of the sand).

(*a*) *Lime mortar*, usually composed of 1 part lime, 3 parts sharp river sand, or 1 part lime, 2 parts sand, 1 part coarsely ground coke.

(*b*) *Coarse mortar*, composed of 1 part lime, 4 parts coarse sand.

(2) *Hydraulic mortar*, characterised by its power of setting under water or at air contact.

(*a*) *Hydraulic lime mortar*, composed of hydraulic lime and sand.

(*b*) *Cement mortar*, composed of cement and sand, usually 1 part cement and 5 parts of sand. Lime may be added.

The lime mortar is cheap, but usually it is mixed with cement for underground use. Portland cement is almost invariably used. It is usually made from a mixture of limestone and clay ground together wet until extremely fine, then dried, calcined and ground again to about the fineness of flour.

The types of lining can be considered under separate headings:

(*a*) *Brick linings*. The brick lining may be straight or arched walling. Straight side walls resist the vertical strata pressure, but they can be given a slight inside curvature to reduce the effect of lateral pressure and distortion of the roadway section. Increasing the wall base bearing area avoids tension in the lining and enables the side walls to withstand a greater vertical pressure. The wall is generally one and a half or two bricks thick, but with larger cross-sections may be three or four bricks. The roof support is usually a steel

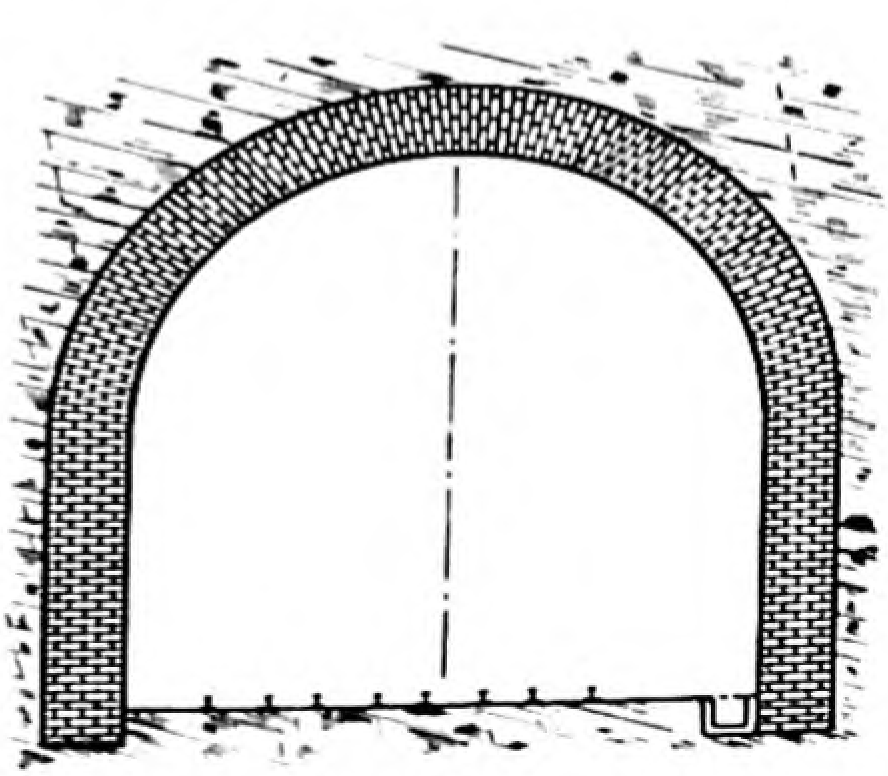


FIG. 162

girder of 'I' section set on the brick side walls. The girders are generally straight, but may be cambered or arched in order to take up bending and compression.

Where the walls have to withstand heavy vertical strata pressures, straight walls cannot be used and the support must be arched. The bricks in the walling are set radially and may be separated by timber pads. The support may be a half-arch set on straight walls, completely circular or open at the base.

The greater the ratio of the height of arch to the diameter, the greater is the resistance to strata pressure; for the half-arch the

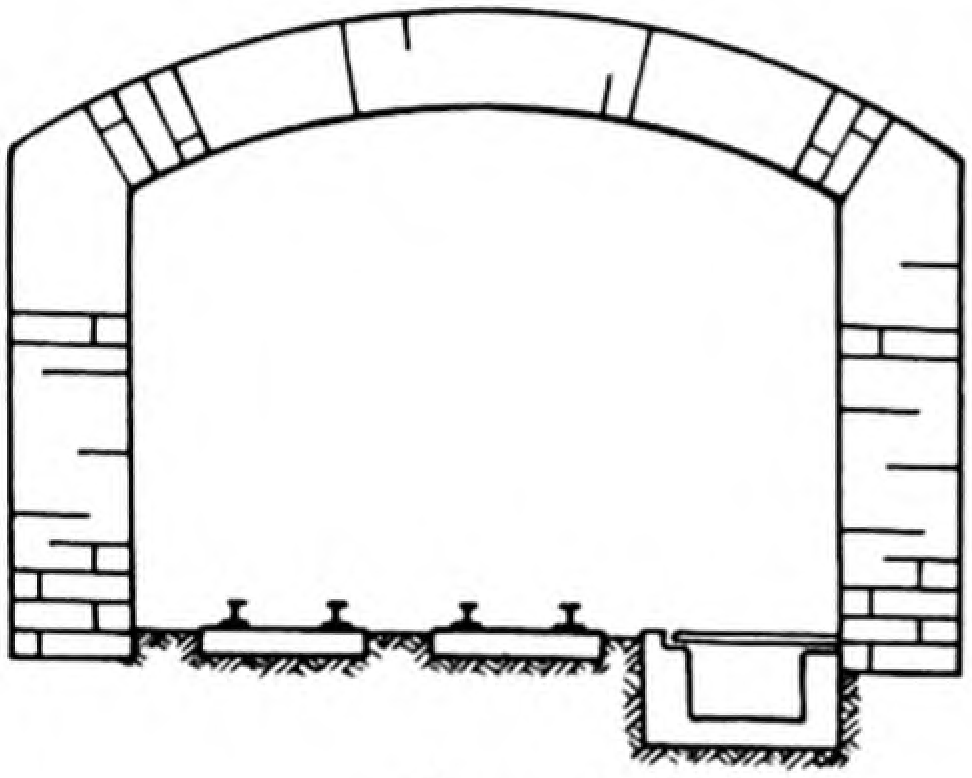


Fig. 163

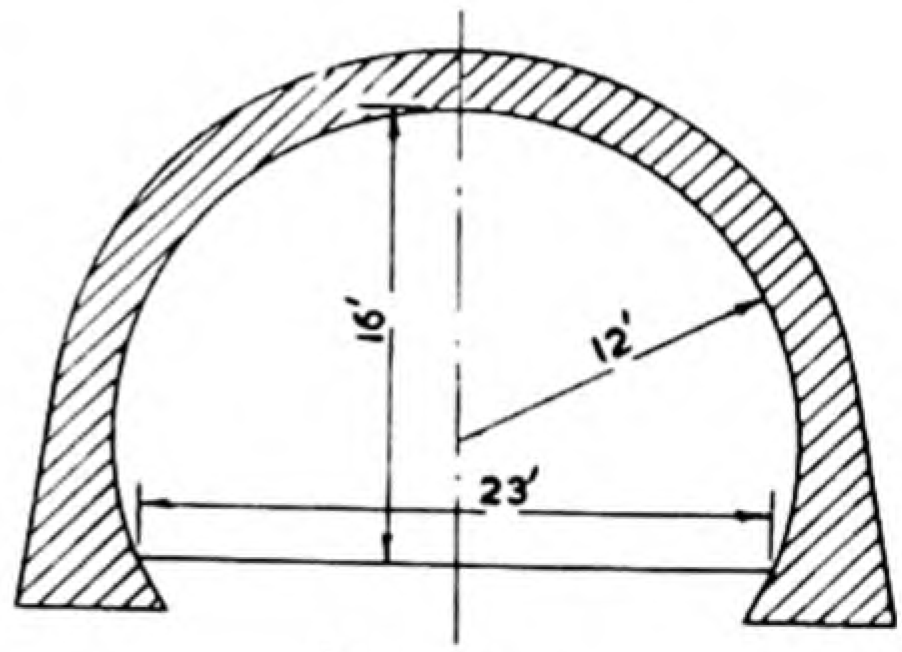


Fig. 164

ratio is 1 : 2. Where the strata pressure is small, a ratio from 1 : 5 to 1 : 10 may be sufficient. A semicircular arching set on straight side walls, serving as a support for a pit bottom, is shown in Fig. 162, while a cambered arch main roadway section is illustrated in Fig. 163. An open circular brick support installed in a shaft bottom is shown in Fig. 164. This form is able to withstand heavier lateral and vertical pressures, while the foot of the wall has been widened to increase the bearing area and reduce the risk of floor 'creep'.

The elliptical section, which conforms most closely to the natural arching of the strata, has been introduced. The disadvantage of this form of section is the large height-to-width ratio, the roadway width being restricted for haulage purposes, since the major axis of the elliptical support should be vertical.

The completely closed circular or elliptical form of support is to be recommended in all cases where the roadway is driven in water-bearing strata. The brick lining should be as tight as possible and built up a short distance from the rock sides, the space behind the

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

which is built up from alternating courses of brick and timber. This form of support is, however, seldom used.

A disadvantage of any form of lining in which timber is used is the danger of fire. It is recommended that the timber should be fire-proofed by impregnation, although it should be remembered that this treatment, while making the timber difficult to burn, does not render it completely incombustible.

(*c*) *Concrete and reinforced concrete lining*. The high compressive strength of concrete has led to its wide introduction as a material for the support of shaft bottoms, main roads and underground engine rooms, but where tension has to be taken, steel reinforcement should be used. Concrete provides a smooth surface to the roadway lining and thus offers less resistance to the air current than steel arch girder or brick lining. Concrete also allows a wide range of road-way sections to be introduced, using shuttering corresponding to the section required.

The low tensile strength of concrete and the difficulties involved in repairing damaged sections are great disadvantages. Concrete of any form should not be exposed to roof movement during the hardening process, since such movement is detrimental to securing the high compressive strength which can be obtained under normal conditions.

The difficulty of repairing is especially important in reinforced concrete. The removal of a damaged section is often a protracted operation involving the use of flame-cutting equipment, which it is not always possible to use. Reinforced concrete, therefore, is used only under special conditions underground.

In the case of concrete support, three types of lining may be adopted: the monolithic lining, the concrete segment lining and the lining formed using precast concrete blocks. The latter form of lining has been found to suffer the least from the disadvantages previously described and is widely used, although less popular than the steel support, which is easier to repair and less expensive.

Monolithic lining. In this form of continuous lining, the concrete is poured, stamped or pressed into position behind shuttering, the lining being from 1 to 2 feet thick. Where stamping is being used, the lower section is set in position followed by the upper and roof sections. The stamping is done by using steel-headed stampers or compressed-air stamping machines. The concrete may also be

vibrated by using vibration devices to obtain maximum strength. Poured concrete is cheaper and simpler, but less resistant to compression loads. In this method, the concrete is pumped behind the shuttering through flexible pressure hose. The method is more suitable for a support which is being installed as a continuous process, such as in shafts where the lining is installed from a curb upwards. It is important that the water-content of the concrete mix is kept as low as possible, as the strength decreases with increased water-content.

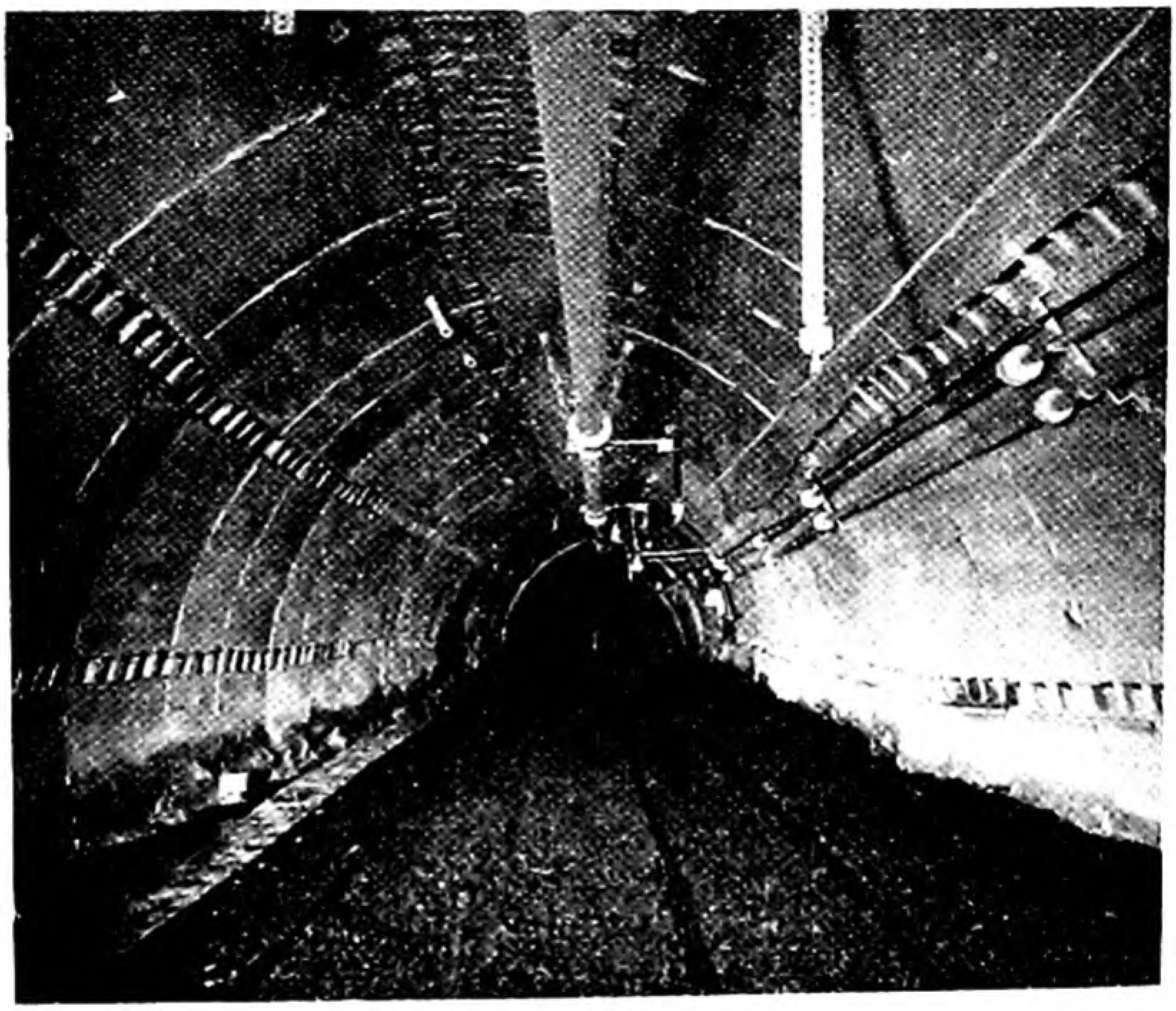


FIG. 166

In installing a lining by the pressing process, the concrete lining is poured in sections behind the shuttering, after which a cement injection is made into the lining. The method is complicated and the cement injection cannot be adequately controlled, and the injection method is used only when 'tying' brick lining to the strata.

Concrete segment support. This form of support is not a complete monolithic construction. The concrete lining is in segments, varying from five to seven, in the roadway section shown in Fig. 166. Each segment is separated by wooden fillers or inserts. The lining is

therefore of the yielding type, the original thickness of the inserts being approximately 3 to 8 inches, the walling being 10 to 20 inches and the concrete mix 1 : 5. Care must be taken in making the wood fillers in order to avoid the creation of high edge pressure at these seams, which result in chipping off of the concrete. As in Fig. 167, the long prop inserts are set vertical to the roadway and short header props are set between each pair, with flat wooden planks resting above the top and below the long prop insert in contact with the concrete segments. It is necessary to provide the flat wooden planks in contact with the concrete to give a more uniform pressure distribution and thus avoiding the possibility of forming fissures in the concrete segments. It may be an advantage to fill in the header

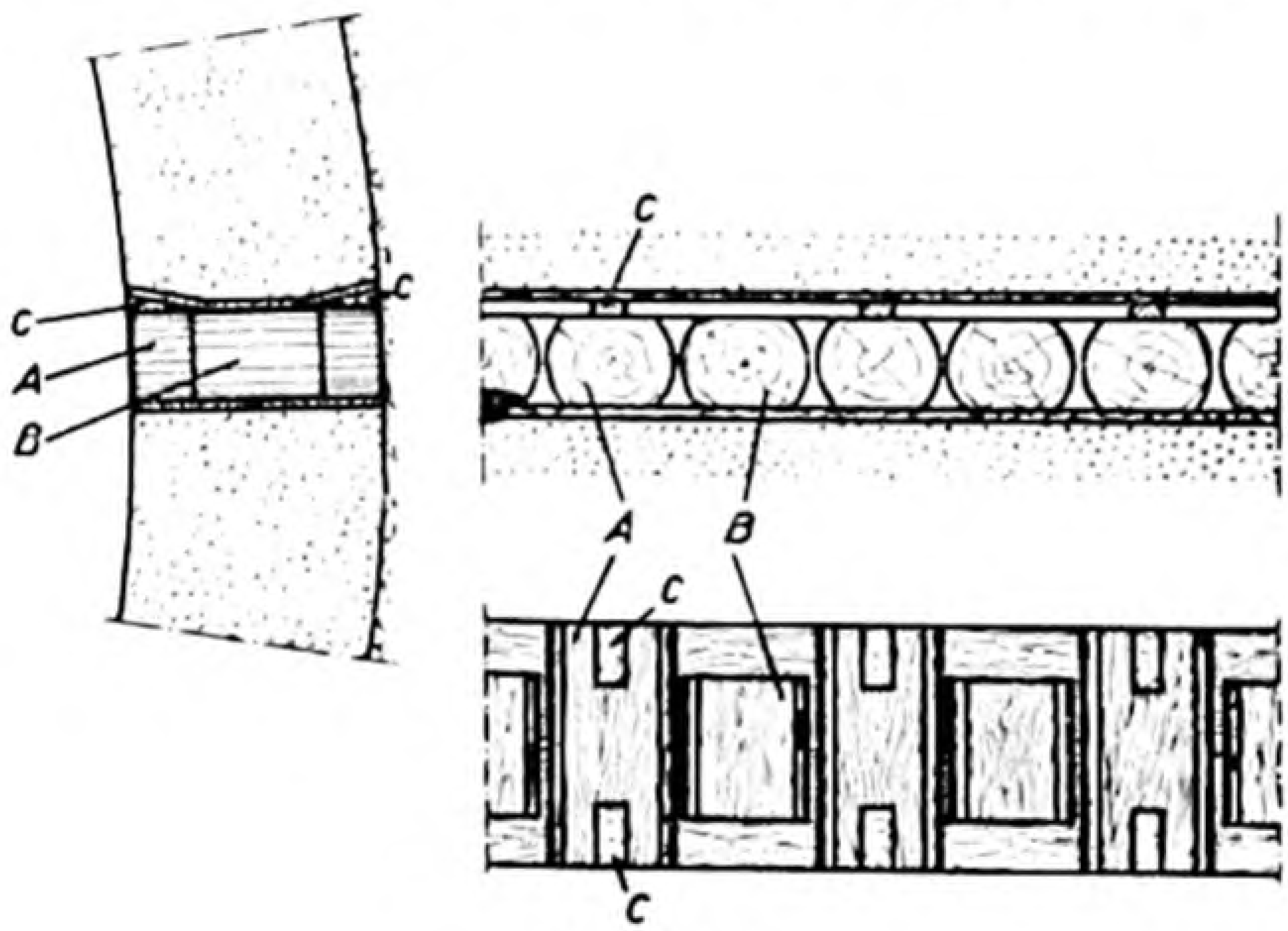


FIG. 167

course with wood and brick, thus providing more than a simple crusher course between segments. This method allows a better opportunity for absorbing the effects of greater pressure near the roof.

Precast concrete block lining. The concrete block form of lining has two main advantages over ordinary brick lining. In the first instance the blocks are larger and can be cast to any shape required for a particular roadway section and, secondly, the precast concrete block has a much higher compressive strength. The compressive strength of the blocks depends upon the care taken in manufacture and varies from 4,200 to 8,400 lb. per square inch. The compressive strength of the precast concrete blocks is higher than that which can be obtained by stamping and, since the hardening process is complete before use,

the compressive strength is unaffected by strata movement. Maintenance for this form of lining is easier and cheaper than for stamped concrete.

The concrete blocks are cast by stamping the concrete in moulds. The moulds can be water-covered, as in the manufacture of the double-wedge blocks in the Herzbruch system, during the first five days of the hardening period. When the water has been drained off, the blocks are removed from the moulds and stored in the open air. After this additional air-hardening process, which lasts four weeks, the blocks have reached their greatest strength and can be used for lining. The compressive strength of the double-wedge concrete blocks in a 1 : 4 mix is 7,000 lb. per square inch, if special attention has been given to the sizing, texture of the sand and gravel used as well as to the ramming of the mixture in the moulds.

Three forms of precast concrete block are in use :

1. Parallel-faced blocks.
2. Radial wedge-shaped blocks.
3. Double-wedge blocks, shaped as a quadrant of a sphere.

In many instances, parallel-faced blocks are being used in conjunction with wedge blocks ; the parallel-faced blocks for vertical walls and the radial wedge blocks for the crown of the arch. Edge pressures have proved to be a disadvantage in practice, since under such pressure the blocks begin to crack at the joints, which leads eventually to complete destruction. The effects of edge pressure can be avoided or minimised by inserting wooden cushion pieces $\frac{1}{2}$ to 1 inch thick between the blocks. If wooden cushion inserts are inserted in layers from 3 to 6 inches thick, as shown in Figs. 168 and 169, it is possible to introduce a yielding effect in the concrete lining.

Because of its shape, the double-wedge block can withstand the heaviest crushing. The block is shaped in the form of a truncated pyramid, and the surface shape depends upon the arch and radius of the roadway to be lined. The same shape is also used when the normal wedge-shaped blocks are adopted.

The opposing inclined faces of the wedge must be considered in order to emphasise their importance in the lining construction. To understand their function, it should be appreciated that the shaped blocks form a ring. This ring may be complete if a floor

arch is incorporated, or open if a floor arch is not used, as illustrated in Figs. 170A and B, 171. The plan in Fig. 172 illustrates how the different rings are placed side by side. One ring consists of blocks with the small face laid on the inside, while the adjacent rings are made up of blocks with the small face to the outside. The rings 1, 3 and 5 correspond to the former, and rings

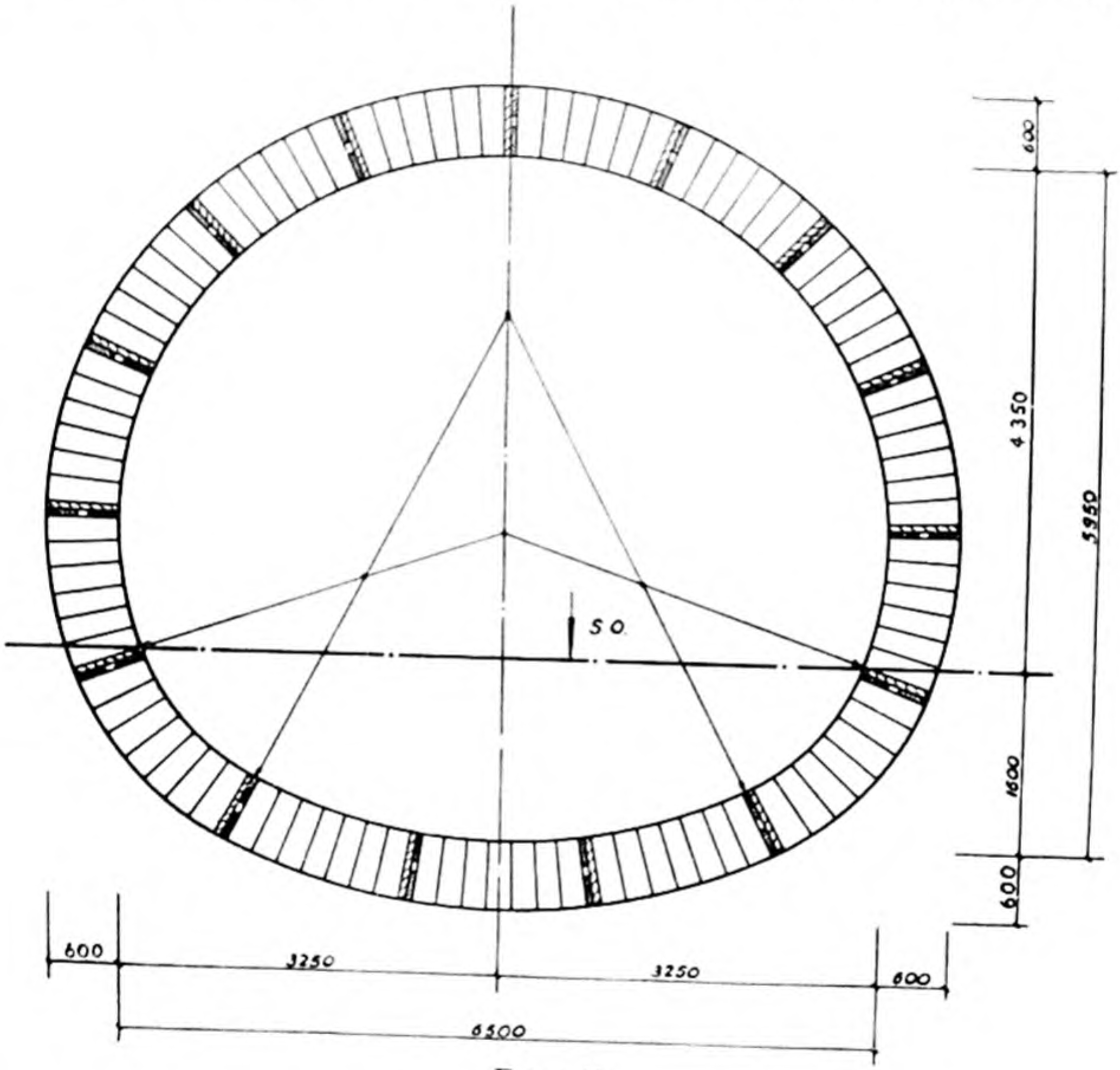


FIG. 168

2 and 4 to the latter. With this side-by-side arrangement, a considerable portion of the radial strata pressure acting on the individual rings is deflected axially and transmitted to the neighbouring rings. The strata pressure is thus resolved into two components of almost equal intensity, one radial and the other axial, and acting at right angles to each other. This resolution of forces, when compared with the usual monolithic lining, results in a considerable increase in resistance against the strata pressure.

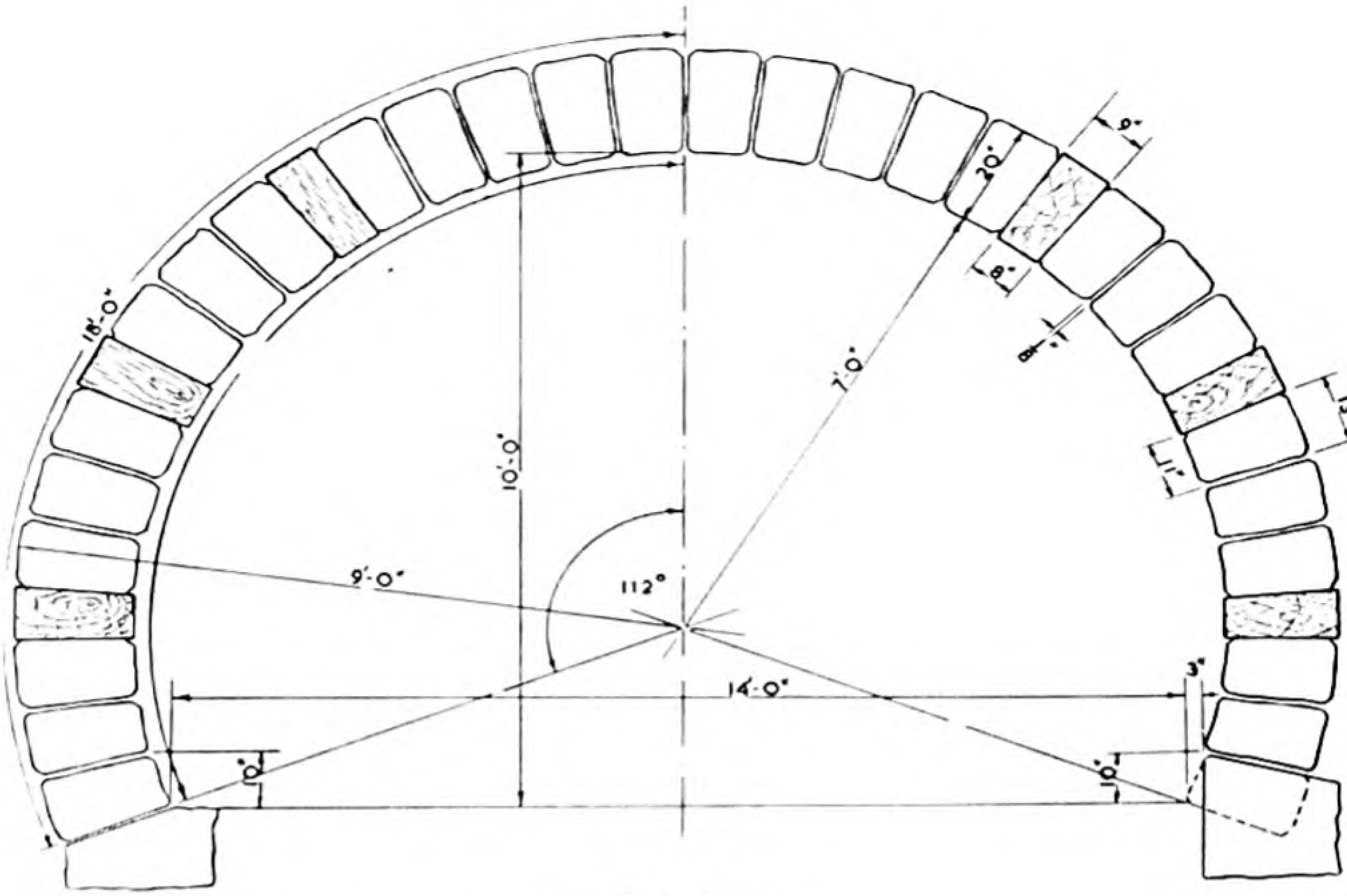


FIG. 169

The thickness of the lining is also very important, i.e. the length of the concrete blocks. Unfavourable results have been produced due to using short blocks, with the result that the thickness of the lining is insufficient to withstand the pressure. The actual length of the blocks varies from 10 to 25 inches, according to the strata-pressure conditions and the finished size of the roadway.

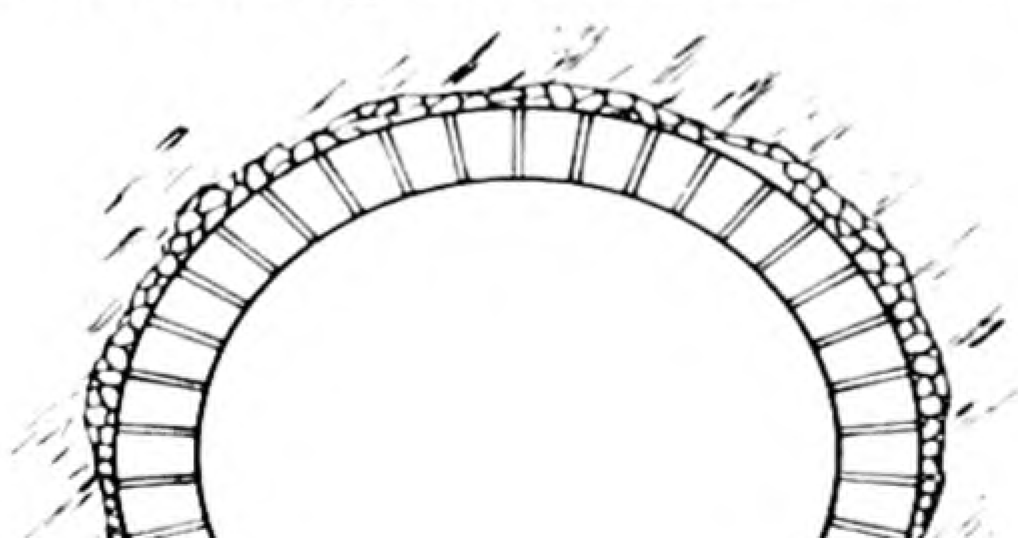


FIG. 170A

The concrete block lining is inserted from scaffolds, as shown in Fig. 173. The scaffolds can be designed as collapsible units, and can be erected so that the centre of the roadway is left free and haulage is uninterrupted when they are in use. Since the blocks are heavy, a small winch is used to place them in position.

As in the case of other lining systems, it is equally important that the lining is fitted tightly to the strata, and it is recommended that

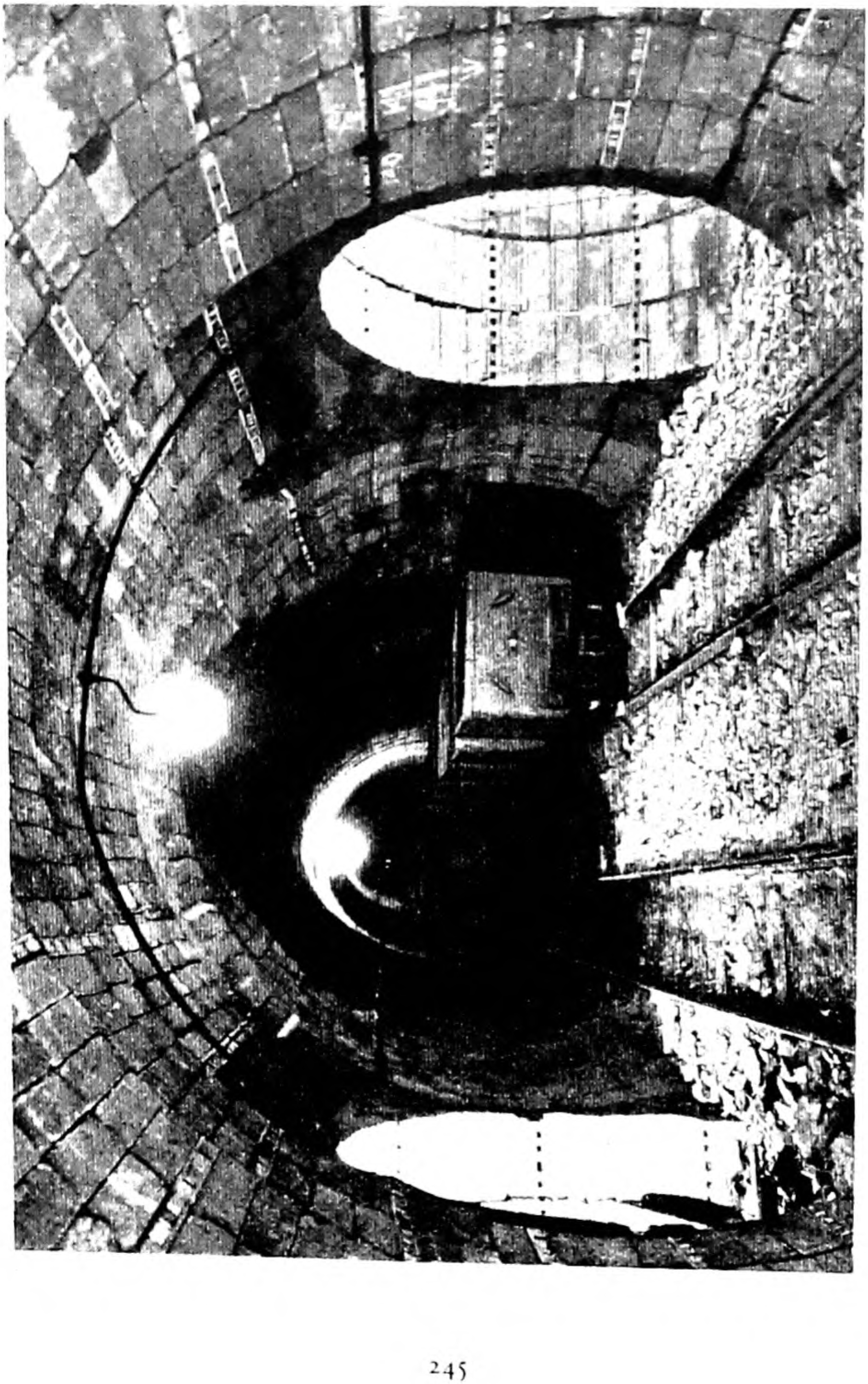


FIG. 170B

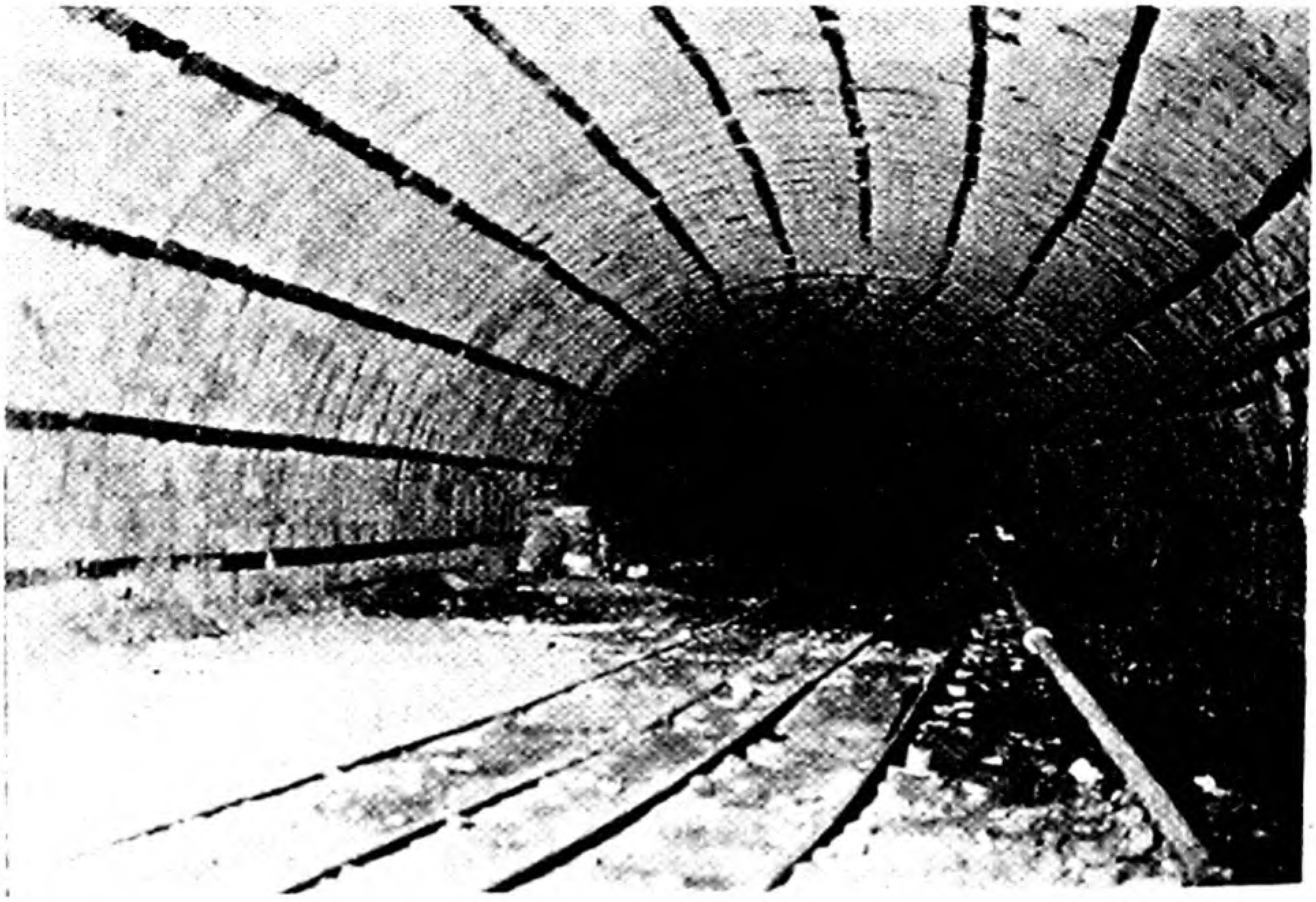


FIG. 171

any space behind the lining is rammed with a back filling of either concrete or waste.

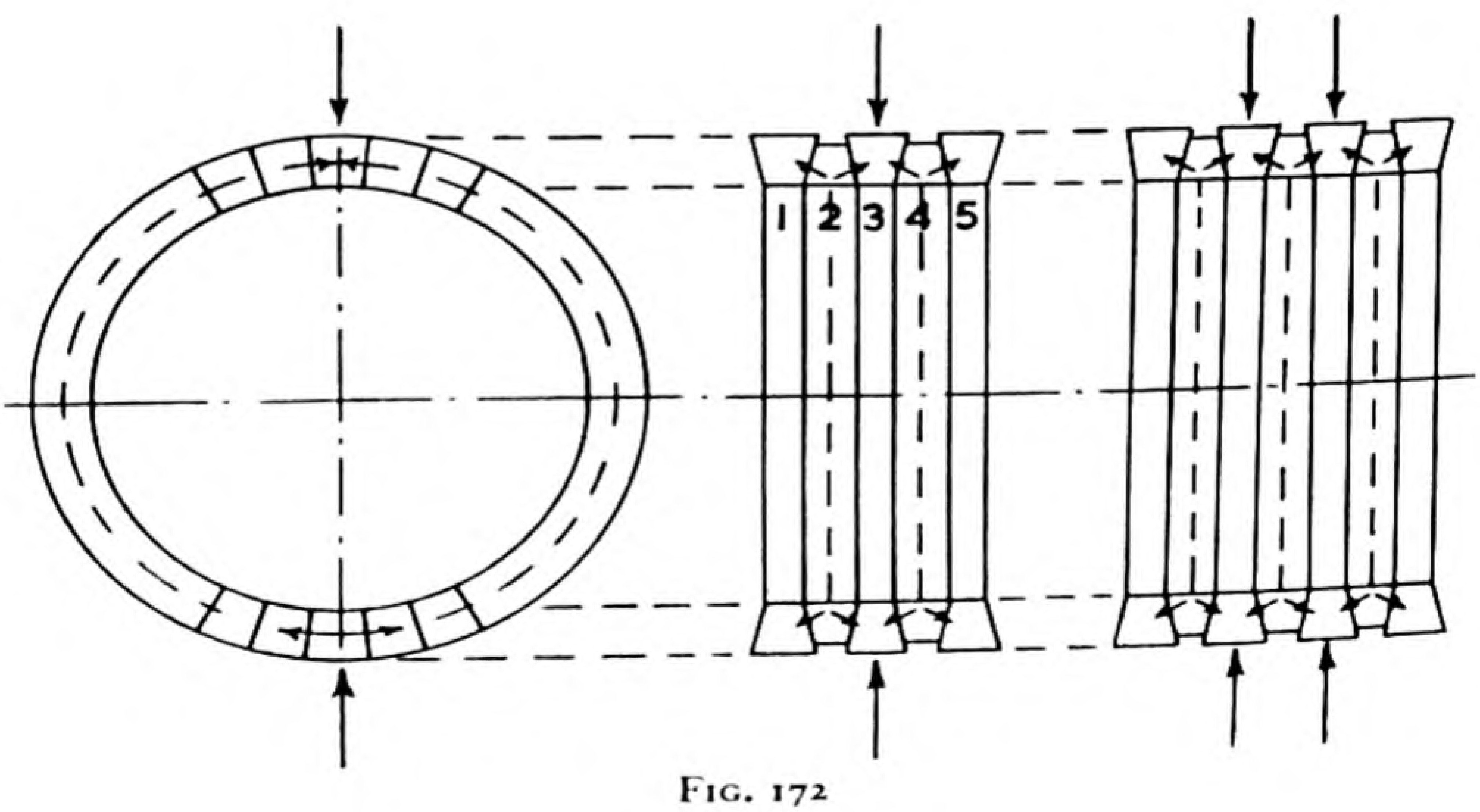


FIG. 172

FIG. 173

PART III

THE CONSTRUCTION OF LARGE CHAMBERS UNDERGROUND

Section 1. *General Remarks*

Large chambers or rooms underground are constructed or driven as part of the actual pit bottom or for the shaft bottom pumping station. Engine rooms for large staple shaft hoists may require a large excavation, whereas locomotive garages, workshops, stores and switch-houses are generally long and narrow and their drivage differs very little from the normal stone drift.

In the construction of a pit bottom, the normal roadway cross-section of from 130 to 160 square feet in the case of main laterals is insufficient and the cross-section required is considerably more. A pit bottom having three tracks for a shaft with a single cage-winding system requires a minimum width of 17 feet, and a height of about 13 feet, giving a cross-section of about 220 square feet. A shaft incorporating two separate winding systems requires a minimum width of about 22 feet, and a height of 17 feet or a cross-section of about 350 square feet. In certain special cases, pit bottoms having a section of 650 square feet have become necessary.

Pumping stations require a smaller space and, considering a chamber accommodating two centrifugal pumps, set side by side, the room must be about 18 feet wide and 28 feet long. If the pumps are set in line, the width of the room required is reduced to about 12 feet, but the length is increased to 50 feet. Where reciprocating ram pumps are installed, double this space is required.

Where the pit bottom is in level strata, it is immaterial when considering strata pressure whether the longitudinal axis of the chamber is parallel or at right angles to the strata inclination. With semi-steep or steep formations, it has been found that rooms set on the dip are less exposed to strata pressure and can be maintained more easily than chambers driven in the strike direction.

(*a*) *The development of large rooms.* The three main methods adopted for the construction of large excavations are as follows:

(i) Immediate construction at the full cross-section,
(ii) The Belgian method, utilising a pilot excavation in advance,
(iii) Initial drivage, leaving a core of rock for final extraction.

The first method is generally preferred, but its application is dependent upon the strength of the strata in which the room is to be constructed. Where the strata conditions are good, there is no objection to this method. The organisation for the drivage differs only very little from normal drivage. Difficulty is usually experi-

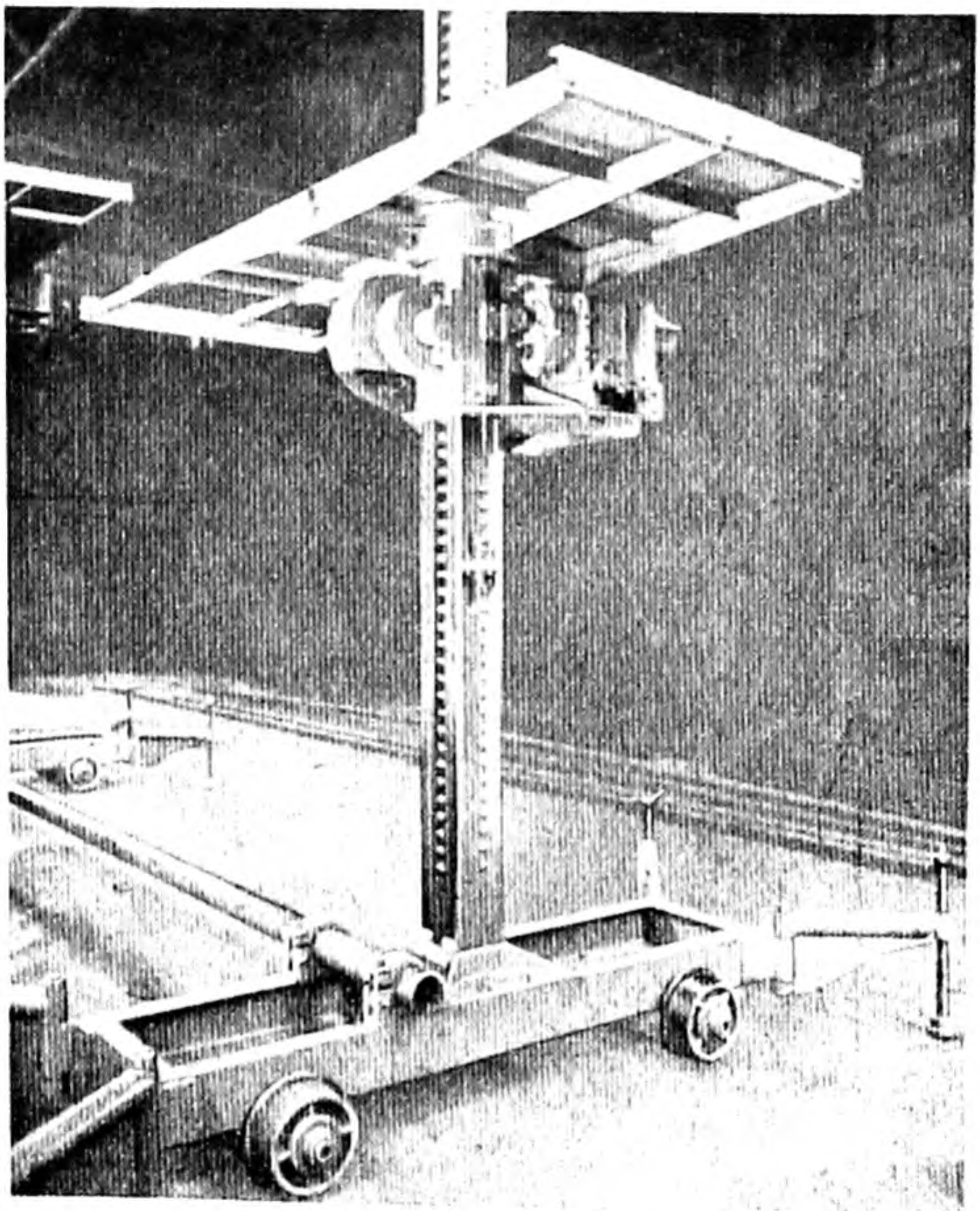


FIG. 174

enced, however, in drilling and firing and setting the supports in a large excavation of this nature. Special platforms must be erected and jacks used for lifting the supports up to the required height.

These operations can be greatly facilitated by the use of a power-driven working and drilling platform as illustrated in Fig. 174. The platform is mounted on a vertical post, which is set on a bogey which runs on rails. The vertical post has a rack which is geared to a pinion on a 65-h.p. motor situated under the working platform. The

motor drives the pinion through a worm gear and is capable of raising 7,000 lb. Heavy loads can be raised on the platform, which can be rotated round the vertical post.

Two identical machines are shown in use in Fig. 175, holding a working platform across the full width of the excavation. Heavy steel arch girders or parts of a scaffold can be lifted by the machine and fastened to the upper part of the vertical column until the arch is set. The working platform can be used to transport materials to the face. It is claimed that, by using such equipment in an excavation

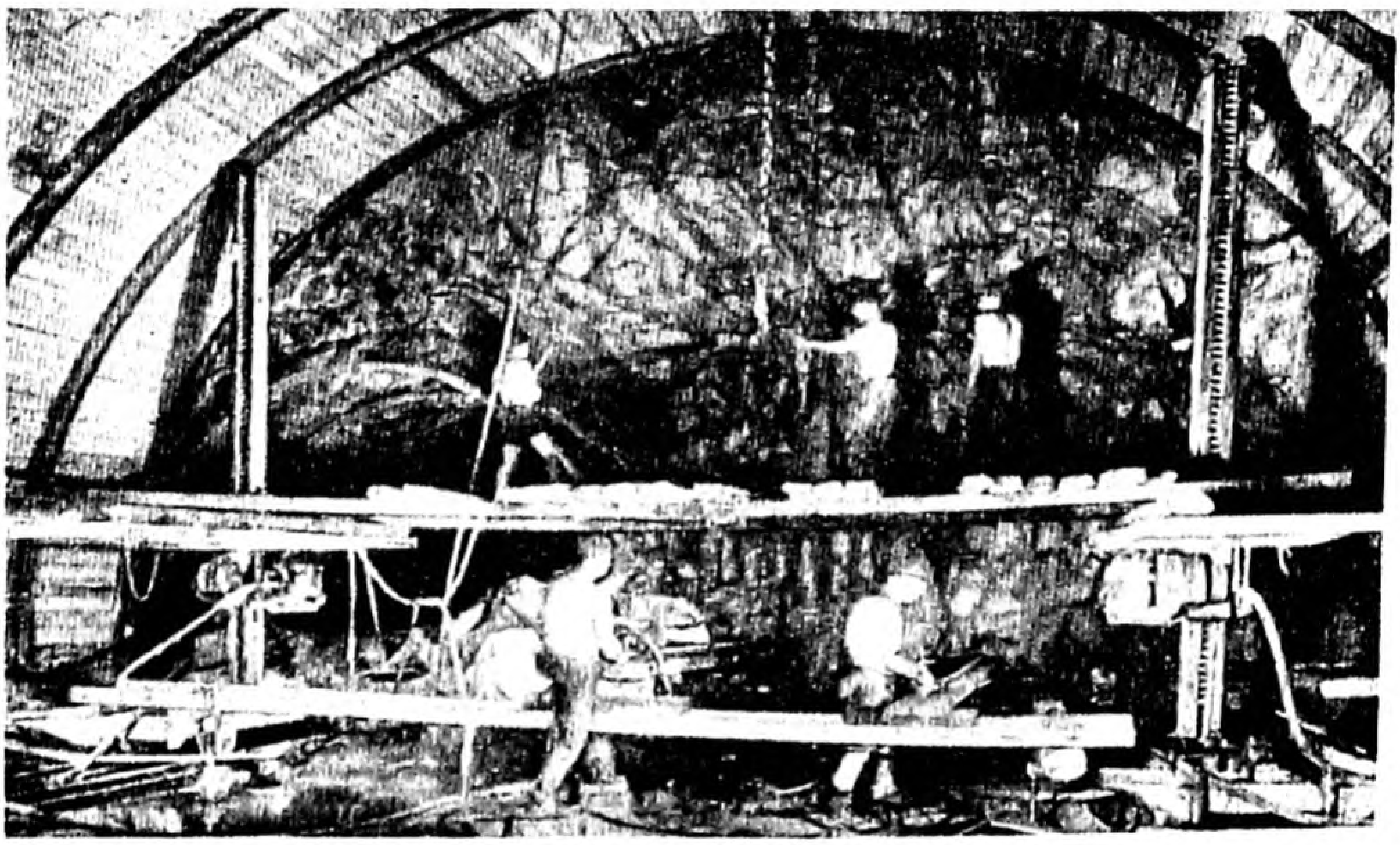


FIG. 175

in sandstone, the performance per manshift has been increased from 45 to 80 cubic feet, while bricklayers have increased their performance from 14 to 28 cubic feet per manshift.

(*b*) *Special methods in weak strata.* The Belgian method of drivage has been preferred where large excavations have been made in weak strata, since the roof is immediately supported in advance of the further excavation work. The sequence of operations used in this method is illustrated clearly in Fig. 176. The method has the advantage of securing the roof above and in advance of the remaining lower section of the excavation. It has the disadvantages that there is the risk of damage to the roof supports by subsequent shot firing in

the bottom section, and it is difficult to get a strong continuity in the supporting back filling with the arch set in advance of the lower section. The method is the commonest for such work, however, in general mining practice.

In order to avoid the difficulties mentioned with the Belgian method, the German core method may be adopted. In this case the lower outline of the excavation is marked by the drivage of lateral roads, which are subsequently raised by top-ripping to the full height of the excavation as shown in Fig. 177. The final side supports are set as the top lifts are taken with temporary supports against the rock core. The roof arch is connected across between the initial

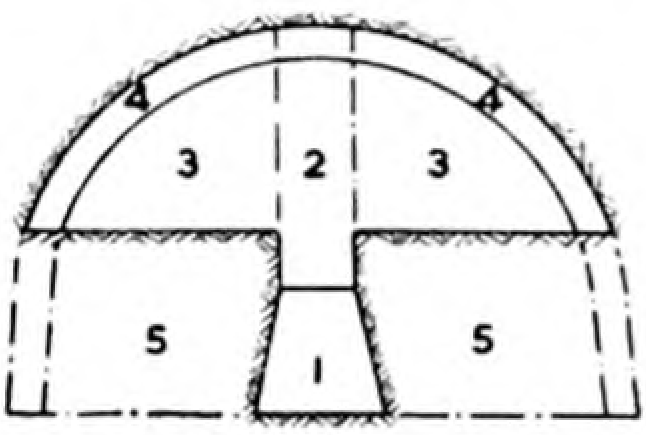


FIG. 176

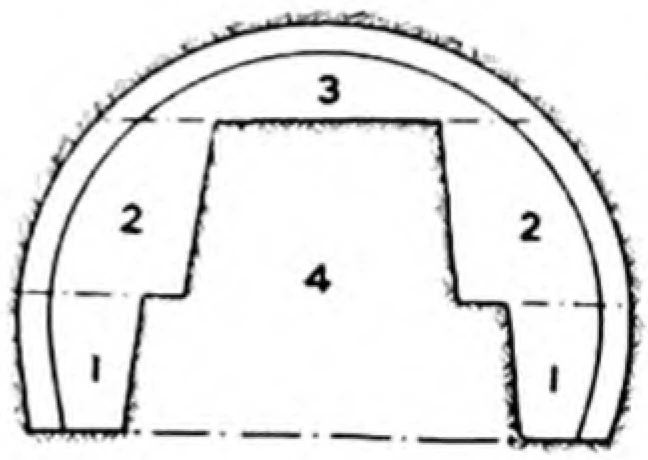


FIG. 177

drivages and the rock core removed by taking bottom lifts. The method combines the advantages of immediate setting of the permanent lining and the excavation is carried out with comparative safety. If care is not exercised, the permanent lining may be damaged when the centre core is removed. The performance in the initial side roads is low due to interference between the several operations in the narrow roadway. Because of this, the method is rarely used in general tunnelling work, but these disadvantages are compensated for by the safety in working when the system is applied to a comparatively short length, as is the case when driving large rooms.

As far as possible, full initial excavation should be used, and where the strata is too weak to be safely and adequately supported with steel arches and close lagging, the Belgian method can be used. The most difficult strata control conditions should be overcome if the centre core method is used.

Section 2. The Support of Large Excavations

(*a*) *General remarks.* In comparison with the support of roadways, large rooms offer no special differences. It should be noted, however, that in view of the importance of the rooms, their long life

and the necessity to reduce maintenance work to a minimum, the supports inserted should be as strong as possible. The following methods of support in their order of importance can be recommended:

1. Brickwork lining.
2. Concrete block lining.
3. Monolithic concrete lining.
4. Steel arch girder supports.

The use of steel arch supports is usually confined to cross-sections up to 250 square feet. With larger excavations, their use is limited owing to the necessity for using heavier girder sections set closer together. The connection with the shaft is difficult with arch girder supports, and a closed form of support is preferred. In this case an adequate key into the rock sides is necessary, as well as care in the choice of materials, mortar and concrete mixes, together with close supervision.

The insertion of wood cushion pieces at the correct places is recommended. Vertical gap construction should be used in the lining, either leaving the gap free or filled with soft wood cushions.

(*b*) *Performance and costs in constructing large chambers.* It has been shown when discussing the methods which may be used to excavate large rooms that the performances achieved and the cost of drivage depend considerably on the strata in which the room is to be driven. Certain methods will allow good co-ordination of each operation and give correspondingly high performances, while other methods in which the disposal of the waste and insertion of supports is difficult, will allow only slow and careful work. In the latter case, the auxiliary equipment required will add considerably to the cost of drivage. Thus, the performance which can be achieved may vary from 35 to 53 cubic feet of rock per manshift, while the performance in support setting will vary from 21 to 42 cubic feet of finished support lining per manshift.

Average costs for excavations under normal conditions are as follows:

Section sq. ft.	*Cost per cu. ft.*
200	10*d.* to 1*s.* 4*d.*
400	1*s.* 2*d.* to 1*s.* 8*d.*
600	1*s.* 6*d.* to 2*s.* 0*d.*

The increase in cost in the case of the larger cross-sections indicates the special difficulties encountered in making such excavations. Where the excavation includes a connection to the shaft and pit bottom, the cost of excavation will increase to about 2*s.* 6*d.* per cubic foot.

The average costs of supporting such excavations are as follows:

Lining	*Cost per cu. ft.*
Brickwork . .	3*s.* 3*d.* to 4*s.* 3*d.*
Concrete block .	4*s.* 8*d.* to 5*s.* 8*d.*
Monolithic concrete .	3*s.* 0*d.* to 4*s.* 0*d.*

Since the average pit bottom cross-section is 200 square feet, the volume of the lining support can be assumed to be about 200 cubic feet per yard, thus the total cost per yard for a pit-bottom drivage, including lining, will be as follows:

Total cost of pit bottom excavation including lining

Lining	*Cost per yard*
	£
Brickwork . .	65–75
Concrete block .	80–90
Monolithic concrete .	60–70

The cost of particularly large excavations may exceed £300 per yard.

PART IV

SINKING AND DRIVING OF STAPLE SHAFTS

Section 1. *General Introduction*

The development of staple shafts can be carried out either by sinking downwards or driving upwards.

The sinking of staple shafts is very similar to main-shaft sinking. The differences introduced in technique are due mainly to the smaller cross-sectional area, the shape (usually rectangular) and the fact that the shorter life of the shaft, which may be from 10 to 20 years, allows lighter supports to be installed. Where the staple shaft

is more important, the section may be circular and the diameter may be as great as 15 feet, in which case the differences between main- and staple-shaft sinking are less.

(*a*) *Shot firing.* The rock is usually blasted, in which case the shot holes are drilled wet with heavy percussive hammer drills. As in drift driving, the bits used can be either hard steel or carbide-tipped bits. The length of the pull or depth of the holes is generally arranged to give as great an advance as possible, and is a maximum the larger the cross-section and the softer the strata. In the case of

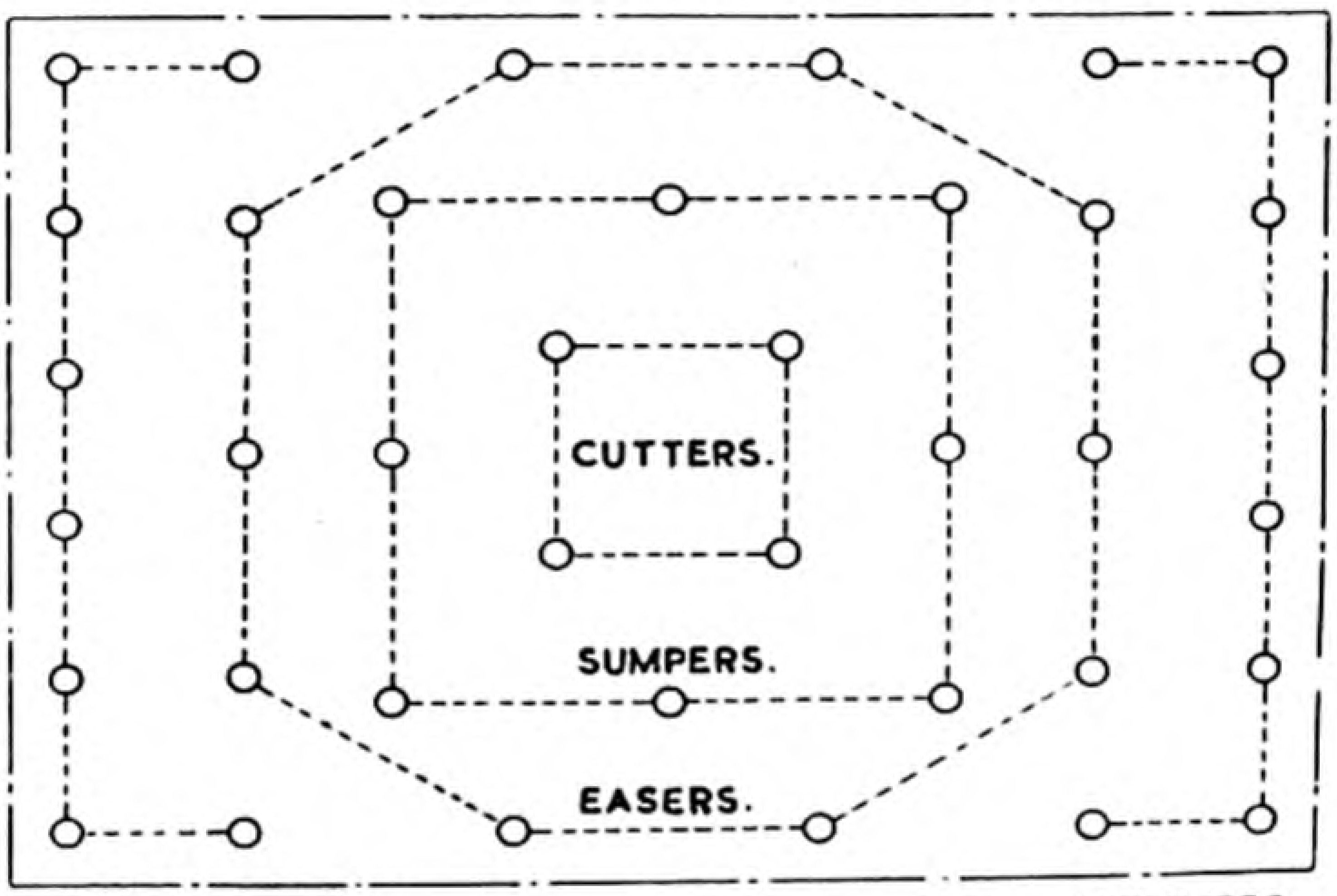


FIG. 178

rectangular shafts having a cross-sectional area of about 100 square feet, a depth of lift of 6 feet is usual. With shafts of circular section, say 200 square feet in area, the depth of lift in soft rock will be from 8 to 10 feet. In very hard rock, in both rectangular and circular shafts, the depth of the lift will be from 4 to 5 feet. The sinking progress can be increased by increasing the lift per round, but consideration must be given to the strength of the shaft sides, which may be weakened by deeper drilling and the use of a correspondingly larger weight of explosive.

The drilling pattern is based on an initial sumping ring, enlarged by a second sumping ring and the outer easer and supplementary holes. Fig. 178 illustrates a drilling pattern for a rectangular shaft 12 feet long by 8 feet wide. The round consists of four cutters, eight sumpers, ten easers and sixteen trimmer or corner holes. The rock

being sunk through is mainly sandy shale and sandstone. The drilling pattern for sinking a round staple shaft is illustrated in Fig. 179. The shaft is 16 feet in diameter, and three centre sumpers with an outer ring of six sumpers are required, followed by twelve easers and twenty trimmers on the outer ring. Gelignite is usually used as an

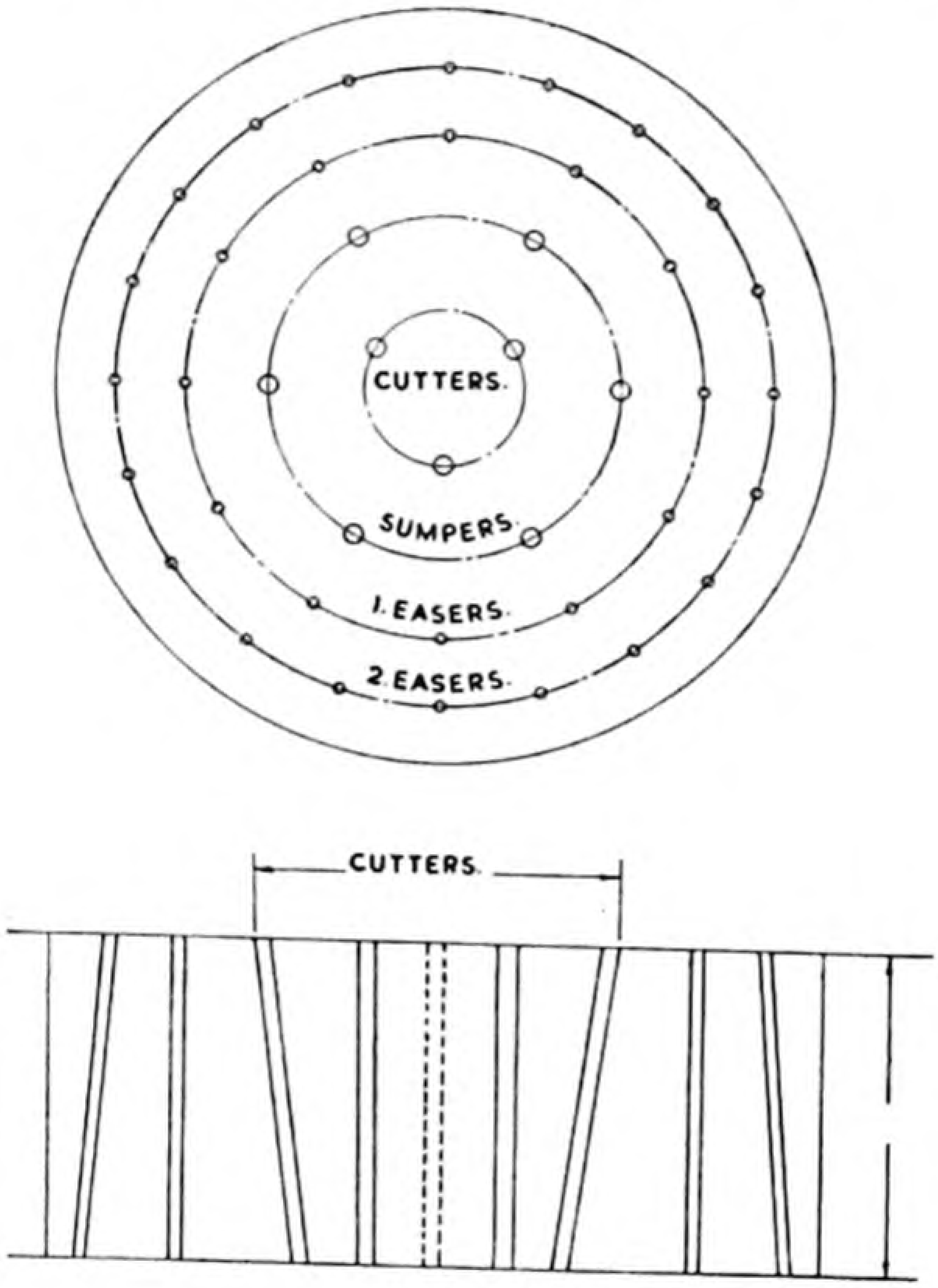


Fig. 179

explosive but, in the case of very hard and solid strata, the more expensive and powerful blasting gelatine is recommended. Where the sinking is being carried out in gassy mines, the weaker permitted explosives must be used.

Electrical shot firing is always adopted with simultaneous firing of the round. The shot-firing cable is wound on a separate drum on the upper level near the sinking hoist and is sufficiently long to reach the final depth of the sinking. Firing is usually carried out in two groups, the centre sumpers being fired first, loaded out and followed by the easers and trimmers.

(*b*) *Shaft plumbing.* Careful plumbing of the shaft is necessary in

order to ensure that the shaft is vertical and maintained at the correct size, so as to avoid difficulty when inserting the permanent supports.

In the case of rectangular shafts, four plumb-lines are suspended at the corners of the shaft, equal distances being set out from the shaft sides and the supports.

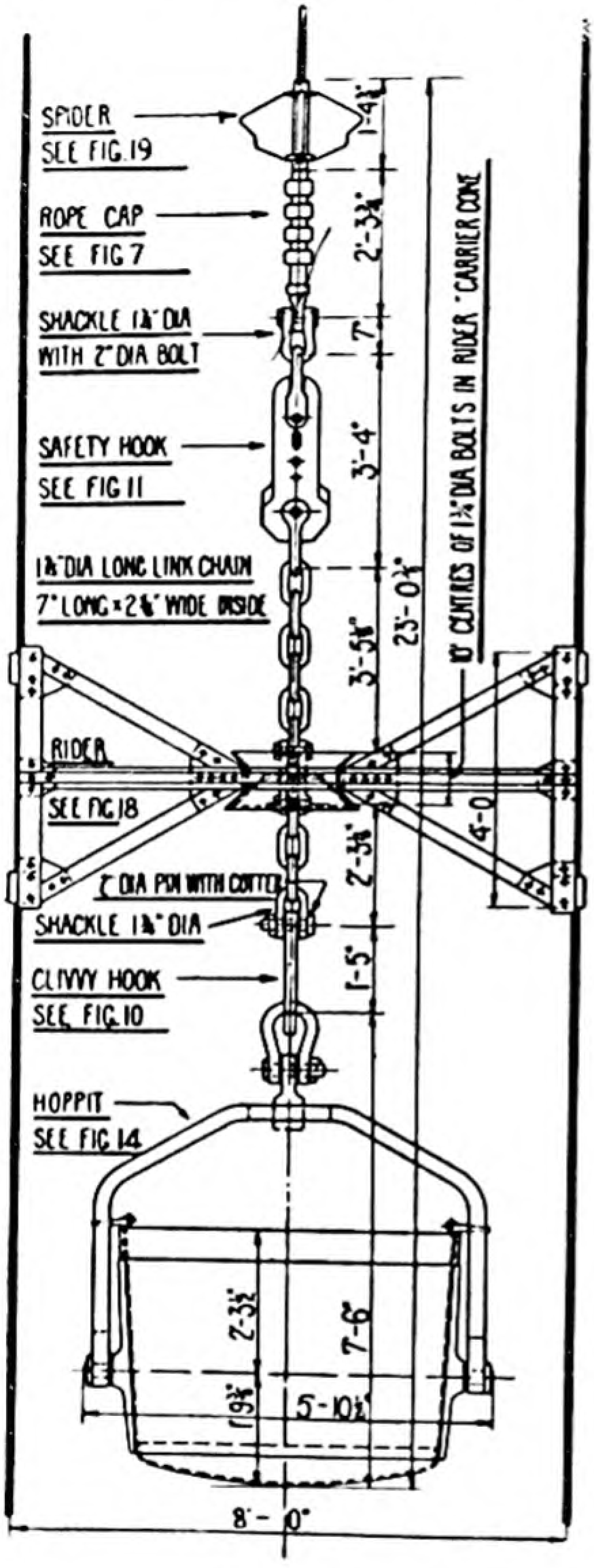


Fig. 180

With circular shafts, a centre plumb-line is used with a radius bar or lathe. The oscillations of the plumb-line can be damped by suspending the plumb-bob in water or oil in a bucket at the shaft bottom.

(*c*) *Hoisting arrangements.* Drum winders are used for the sinking and, in the case of shafts of small sectional area, one hoppit or kibble is used. Where larger shafts are being sunk, two hoppits can be used, one remaining at the bottom to be filled, while the other is being raised and emptied at the upper level. With this arrangement, it is necessary to use a detaching clivvy hook on the rope. In Continental shaft-sinking practice, a 'bobbin' hoist with flat ropes is frequently used. The claim for flat ropes is that they are more free from the liability to twist than round ropes. No rope adjustment is required as the depth increases, and this system is given first choice in shaft sinking. The sinking hoist may be coupled to a polyphase induction slip-ring motor of from 30 to 100 h.p. Compressed-air drive, which is extremely expensive, would be used only where there is no electrical supply.

The sinking kibble or hoppit is usually of steel construction and may have a capacity of 30–40 cubic feet. The kibble is fastened to the

rope by means of a spring hook (clivvy hook) attached to a shackle from the rope socket, the spring hook being attached to a steel bow on the kibble, as in Fig. 180. The steel bow must project beyond the edge of the kibble so that the hook can be unhooked and fastened easily. Discharging of the kibble is done at the tipping stage on the

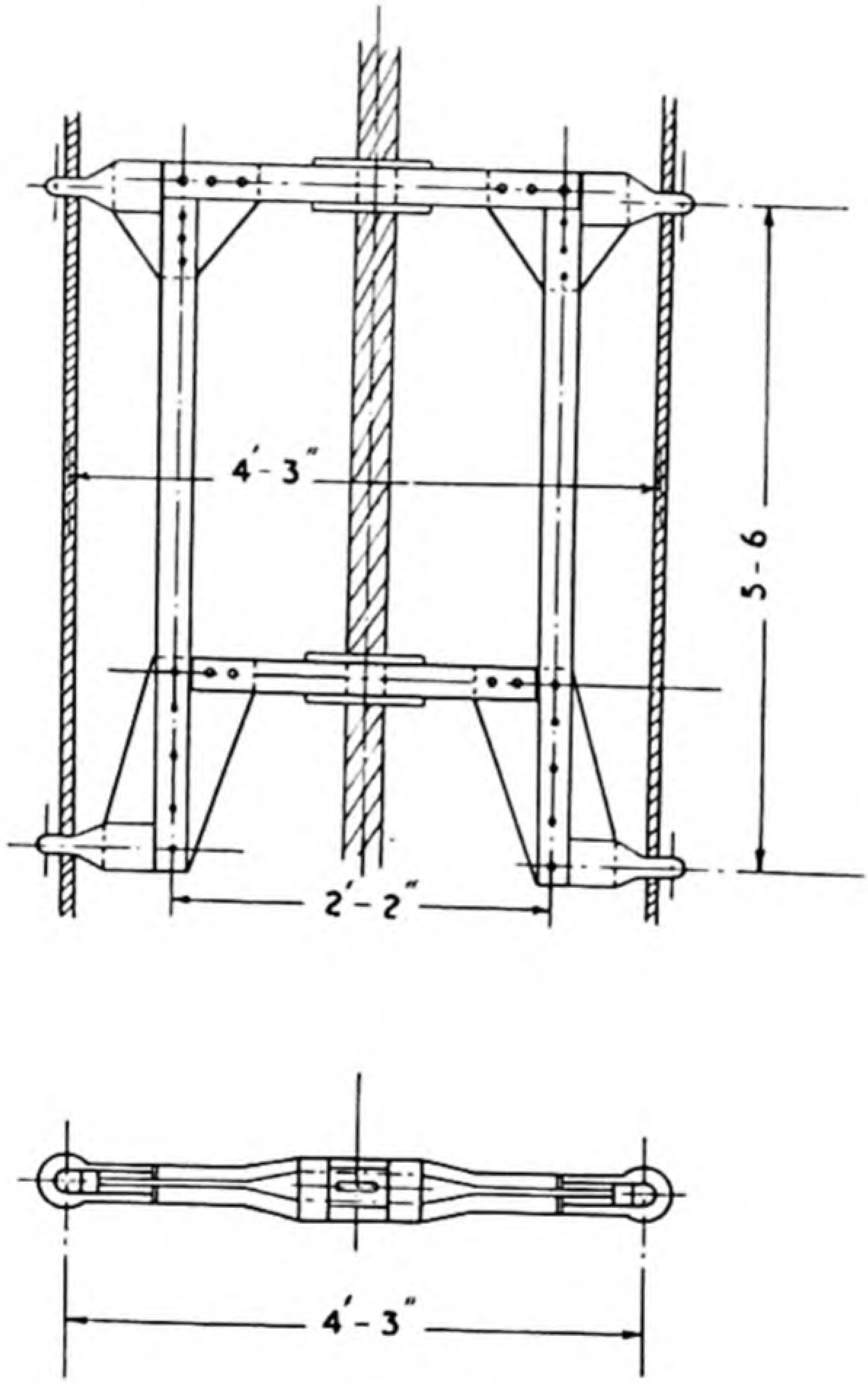


Fig. 181

upper level by putting a chain into a ring on the bottom of the kibble and lifting up the slack rope. The kibble can also be fitted with a chain and ball, the ball catching in a notch in the chute above the tipping stage so that lowering away the slack rope tips the waste into the chute and hopper from which it is taken away in mine cars or tubs.

Where shafts over 100 feet deep are being sunk, precautions must

be taken to prevent the kibble from swinging in the shaft during hoisting, with the possibility of fouling the sides and tipping in the shaft. This is carried out in practice by guiding the rope, and thus the kibble, with a guide skid which follows the kibble during its descent. Where the shaft supports and the cage guides are being installed as the sinking proceeds, as is the custom with rectangular shafts, the guide skid has lateral slippers, or guide shoes, engaging on the fixed guides, as shown in Fig. 183A. Where the shaft supports are not installed and no fixed guides are available, guide ropes are used with another form of rider shown in Fig. 181. The guide ropes, as in Fig. 183B, are hung from pulleys on the upper level and can be lengthened as the sinking proceeds. The ropes are fastened near the shaft bottom to a tensioning device consisting of a steel or wooden framework securely fastened to the shaft side, *vide* Fig. 182. This arrangement allows the framework to be released when lengthening the rope to a lower level. The framework is retained at a point not more than 90 feet above the sinking-shaft bottom, and as men also are being wound and the kibble rides free from the guides, this distance should be kept to a minimum. The guide skid is retained above the tipping stage at the upper level so that the kibble can be tipped freely. This is usually done by the on-setter at the stage level, who inserts a clamp under the guide skid when tipping and removes it when the kibble is ready to descend.

The normal arrangement of head frame and pulleys used at the surface is substituted for an easier arrangement when sinking staple shafts underground. The pulley wheel framework is set in an excavation from 30 to 50 feet in height above the position of the staple shaft to be sunk. Figs. 183A and B illustrate the general arrangement. A platform built from 'I'-section girders takes the sheave pulleys and the tension arrangement, below which is a safety platform. About 20 feet above the floor of the level from which sinking is to start, the tipping platform is installed, beside which can be seen the sinking hoist, guide-rope drums and guide pulleys. The guide pulleys are necessary to take the rope from the hoist and the guide-rope drums (straining drums) to the hoist pulley platform. Sinking doors (Galloway doors) are installed at the upper level and usually are hand-operated to allow the kibble to pass through into and out of the shaft.

(*d*) *Ventilation*. Up to a depth of from 80 to 100 feet it is possible

to sink the shaft without installing means for auxiliary ventilation, since the normal working of the kibble in the shaft will provide sufficient down-draught to keep the shaft bottom clear. With deeper

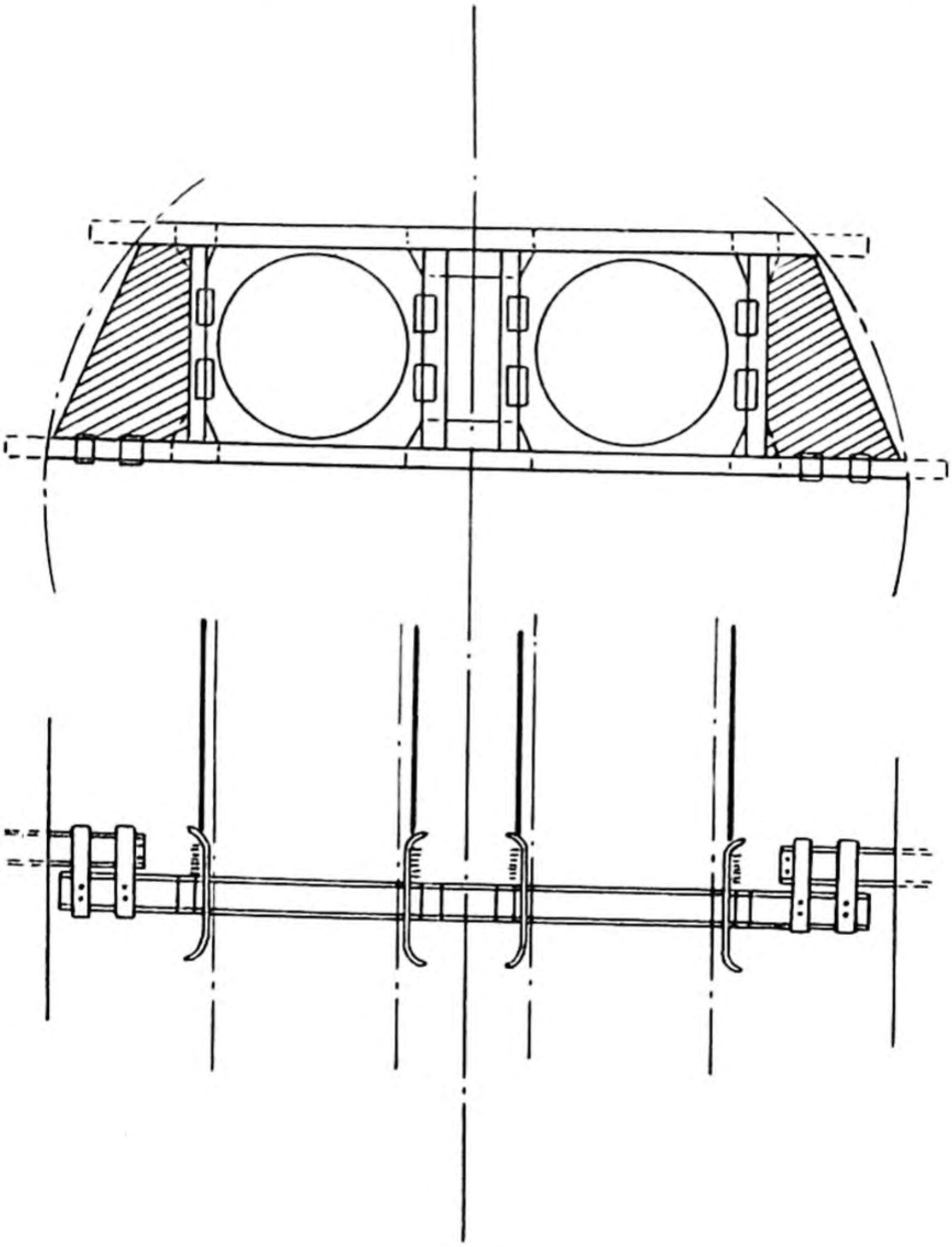
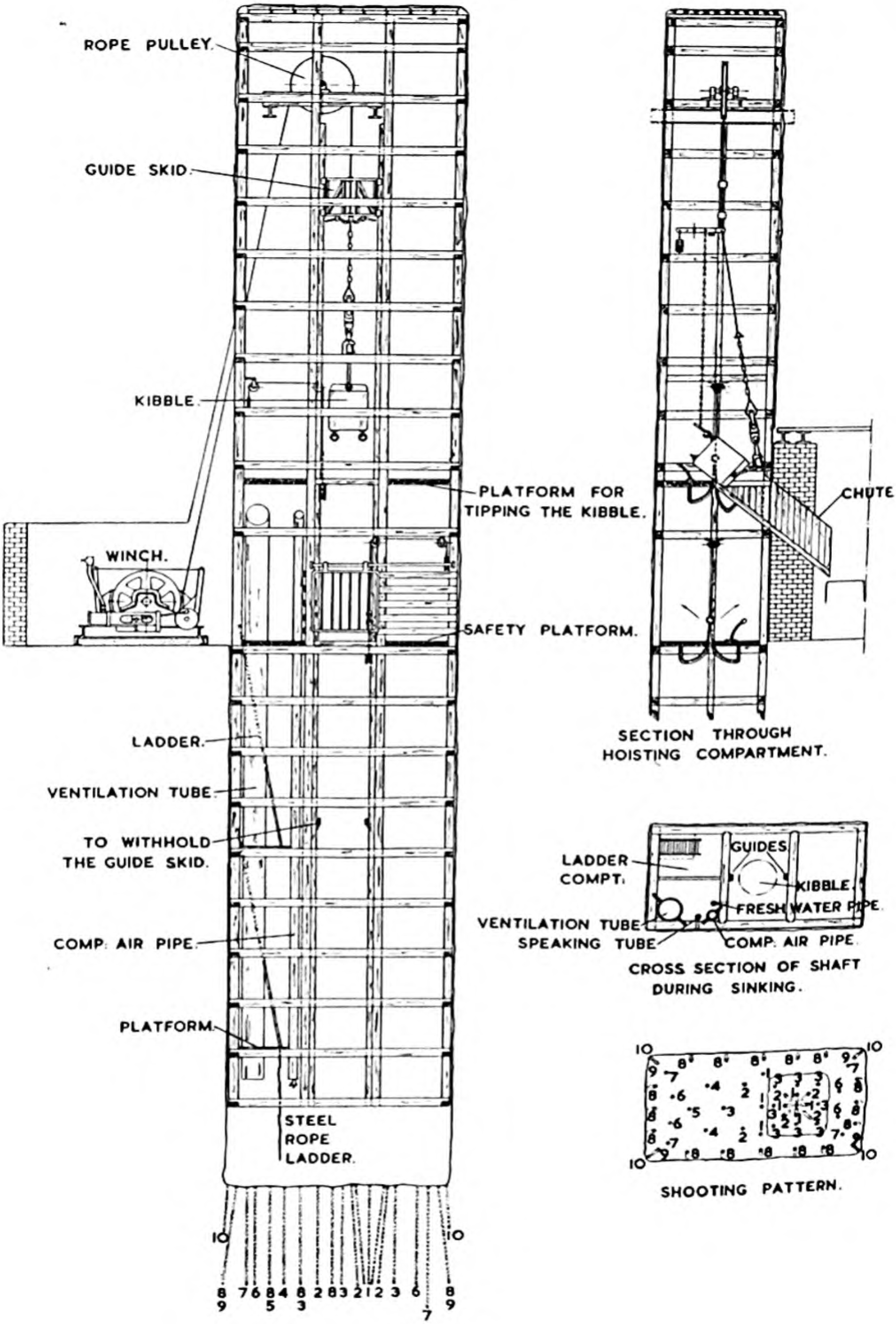


FIG. 182

shafts, ventilation tubes have to be installed or, where practicable, a simple compressed-air jet device, but generally a fan is required. The fan can be either a forcing or an exhaust fan, the former being preferred because of the better cooling effect upon the sinking crew at the shaft bottom, but if there is a possibility of noxious gas at the



ROPE PULLEY.
GUIDE SKID.
KIBBLE.
WINCH.
PLATFORM FOR TIPPING THE KIBBLE.
SAFETY PLATFORM.
LADDER.
VENTILATION TUBE.
TO WITHHOLD THE GUIDE SKID.
COMP. AIR PIPE.
PLATFORM.
STEEL ROPE LADDER.
CHUTE
SECTION THROUGH HOISTING COMPARTMENT.
LADDER COMPT.
GUIDES
KIBBLE.
FRESH WATER PIPE.
VENTILATION TUBE
SPEAKING TUBE
COMP. AIR PIPE.
CROSS SECTION OF SHAFT DURING SINKING.
SHOOTING PATTERN.

FIG. 183A

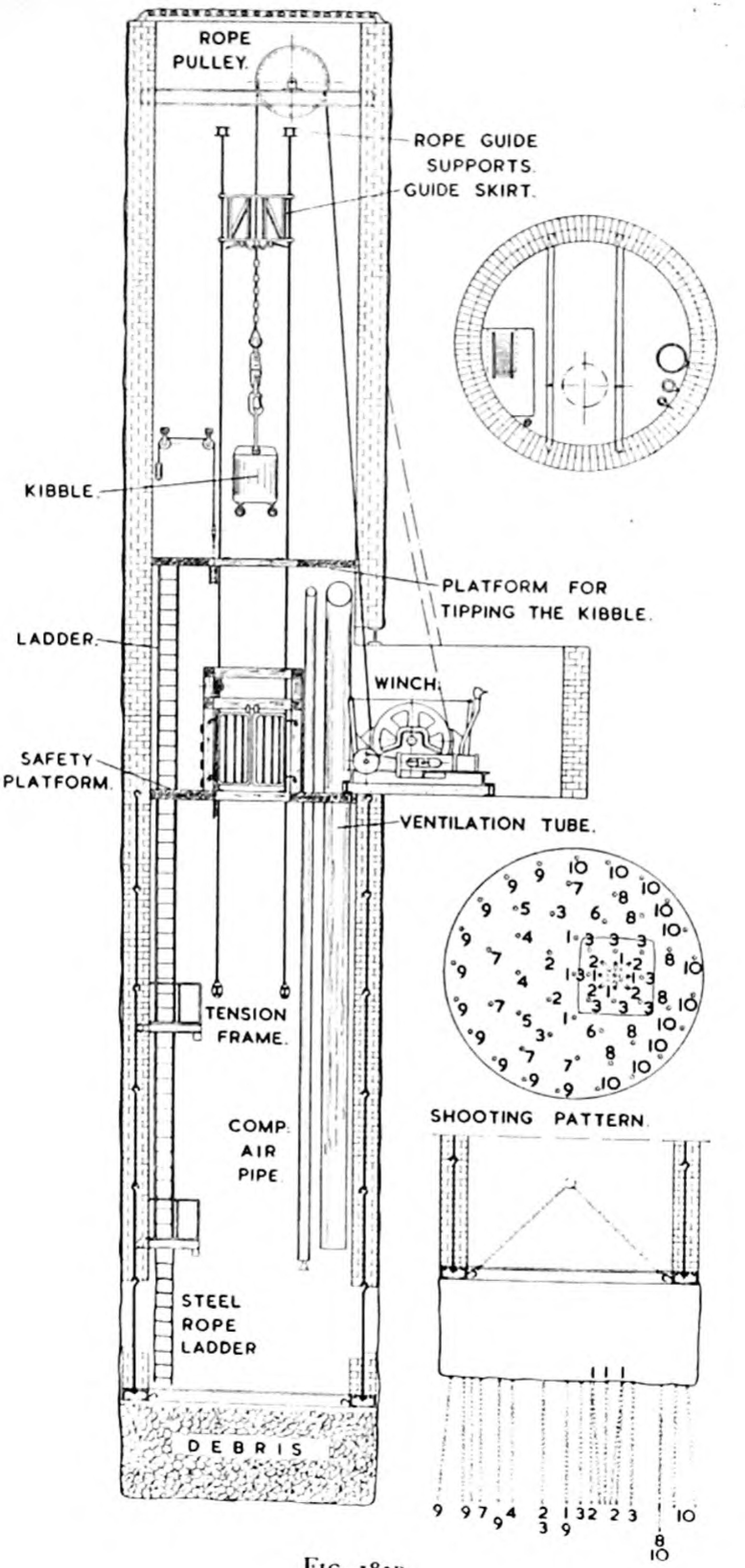

ROPE PULLEY.
ROPE GUIDE SUPPORTS.
GUIDE SKIRT.
KIBBLE.
PLATFORM FOR TIPPING THE KIBBLE.
LADDER.
WINCH.
SAFETY PLATFORM.
VENTILATION TUBE.
TENSION FRAME.
COMP. AIR PIPE.
STEEL ROPE LADDER
DEBRIS
SHOOTING PATTERN.

FIG. 183B

shaft bottom, the latter is advisable. For the same reason, the exhaust fan may be preferable where there is difficulty in clearing out fumes after shot firing. In most cases sheet-metal flanged ducts are used, the diameter of which will depend upon the shaft section and the depth of the shaft. The duct usually has a diameter from 10 to 20 inches. The ducts can be fastened to the permanent supports, if these are installed as sinking proceeds, or fastened with collars and suspension chains from steel brackets fixed in the shaft side. The tubes can be supported from the upper level by means of steel ropes to which the tubes are clamped, lengthening of the ducts then being accomplished by lowering the ropes and adding another tube. At the bottom end of the steel tubing, canvas tubing may be used which can be removed and replaced easily when shot firing.

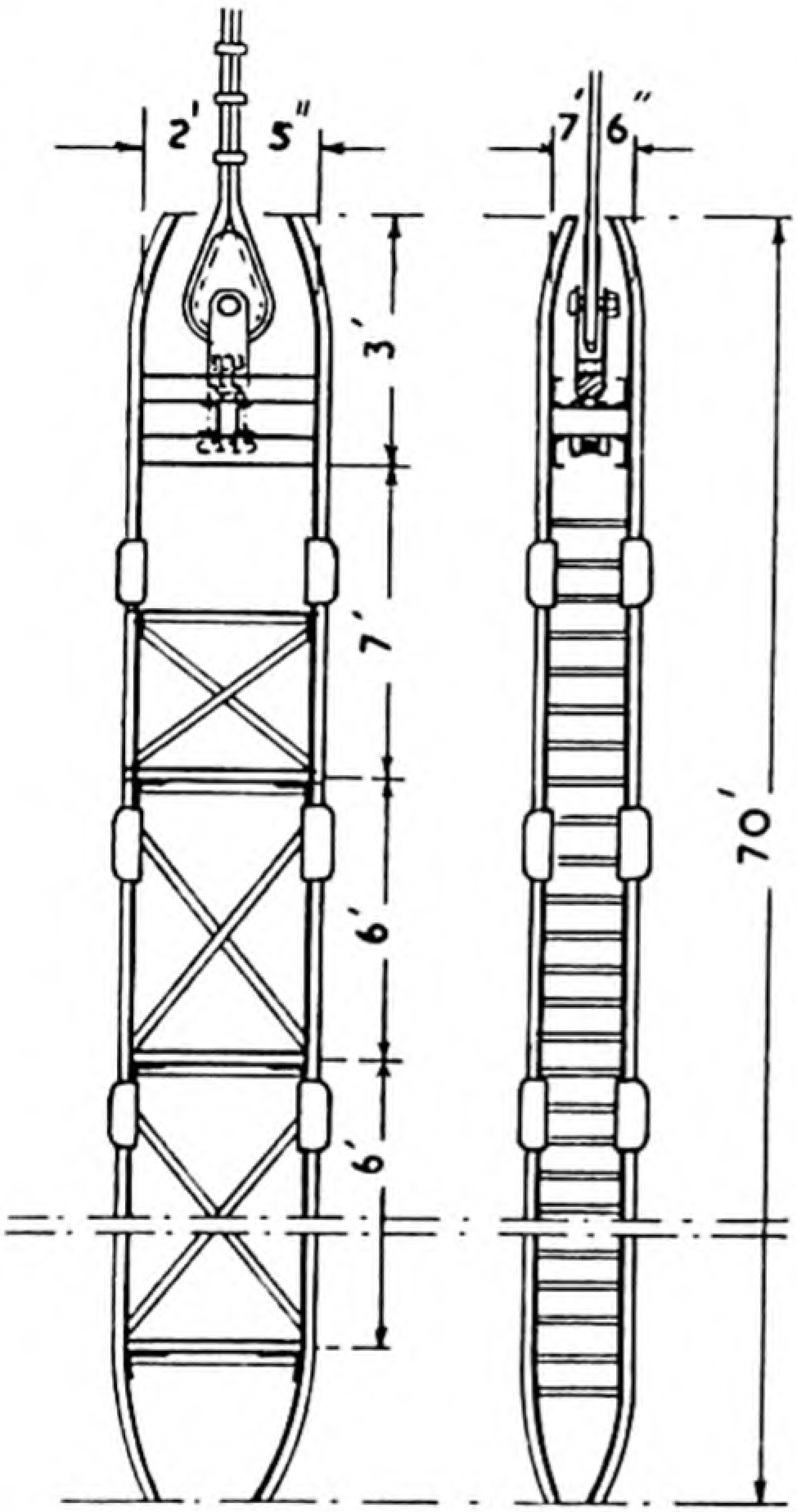


FIG. 184

(*e*) *Lighting*. Ordinary portable lamps are insufficient when shaft sinking and more effective lighting is essential. Large accumulator lamps or compressed-air turbine lamps may be used, but it is better to have main-operated units suspended from a cable which can be hung where they are required. The lighting cable is wound on a drum at the upper level so that the lamps can be raised and lowered during the progress of the sinking.

(*f*) *Man-riding*. The transport of the men to and from the shaft bottom is done normally with the kibble. An alternative means of egress is essential when shaft sinking, to guard against failure of electricity supply, inrush of water, or a pressure burst in the shaft. If the permanent supports are installed within a comparatively short

distance from the shaft bottom the difficulty is easily overcome. Ladders are installed in the shaft in the ladder compartment down to the last set of supports, to which point a rope-ladder can be used from the shaft bottom. If the permanent supports are not being installed, the steel ladder, shown in Fig. 184, having a number of platforms capable of holding two or three men, may be used. The ladder is suspended on a steel rope from a cable winch on the upper level so that it can be raised to the level with the men in the event of an emergency.

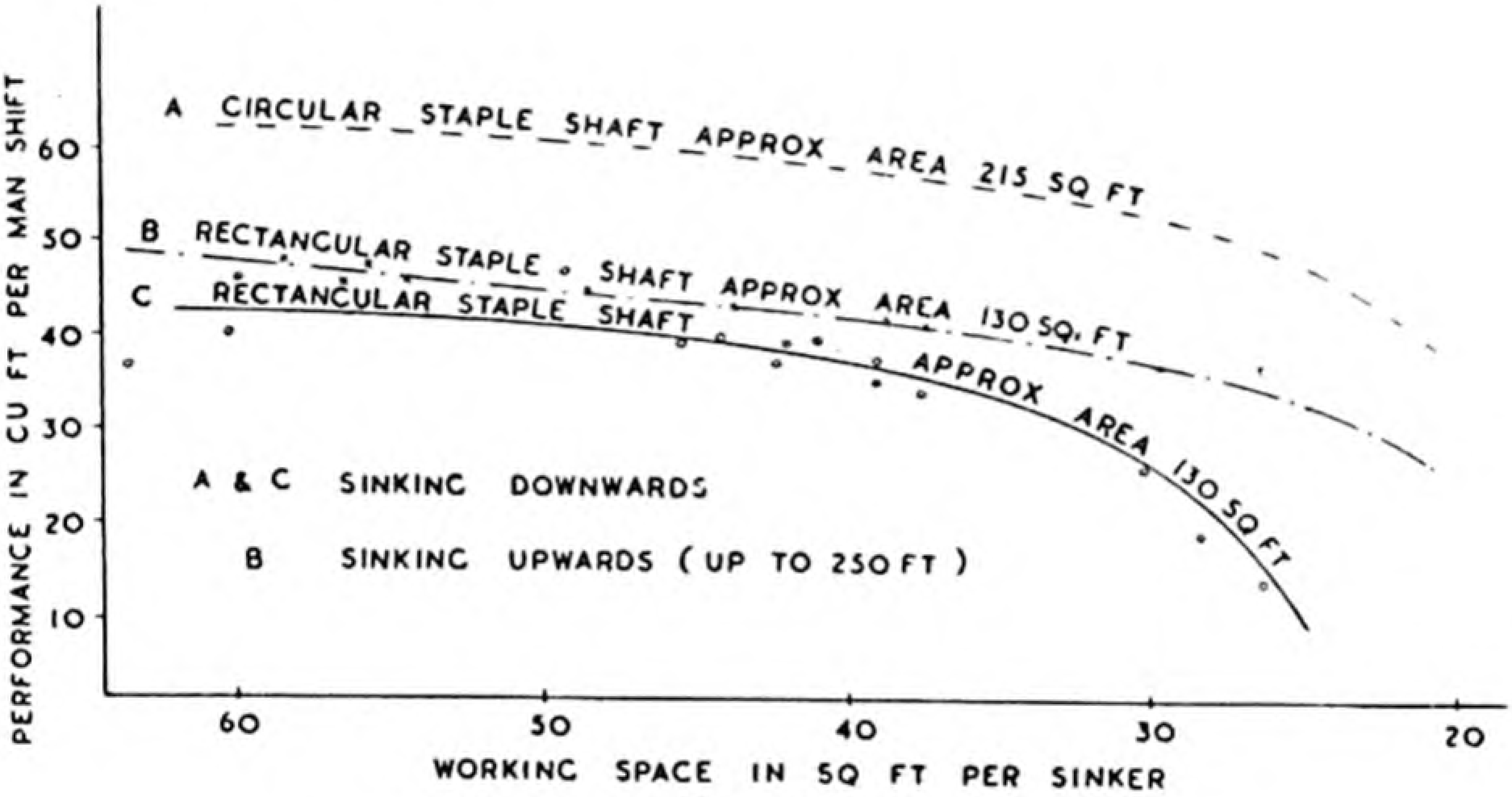


FIG. 185

Section 2. Performance and Organisation in sinking Staple Shafts Downwards

(*a*) *Introduction.* A clear distinction is necessary between the sinking progress per month and the sinker performance, based upon actual shaft bottom (or face) shifts or the total shifts worked on the shaft. The performance of the sinker depends upon the size of the working space allotted to him as well as on the area of the excavation. In the case of a large working space per sinker—that is, with less men per shift—the performance per man will be greatest and it will decrease with an increasing number of shaft-bottom men. The increased number of men tend to hinder each other in their work and, therefore, they are not fully employed over their working shift. Fig. 185 shows a graphical representation of the performance per man-shift in staple shaft sinking in relation to the working space per sinker. The areas considered are approximately 215 square feet

(circular) and 130 square feet (rectangular). With a rectangular shaft of 130 square feet sectional area, the best sinker performance of 1·6 cubic yards per manshift will be reached with a working space per sinker of 65 square feet. The performance is still 1 cubic yard per manshift with a working space of 34·2 square feet but, thereafter, it decreases considerably. In the case of a large shaft area (approximately 215 square feet) the trend of the curve is similar, but a higher sinker performance of 2·1 cubic yards per manshift is obtained with a working space of 65 square feet and only falls away rapidly after a working space of 30 square feet is reached. With a high rate of

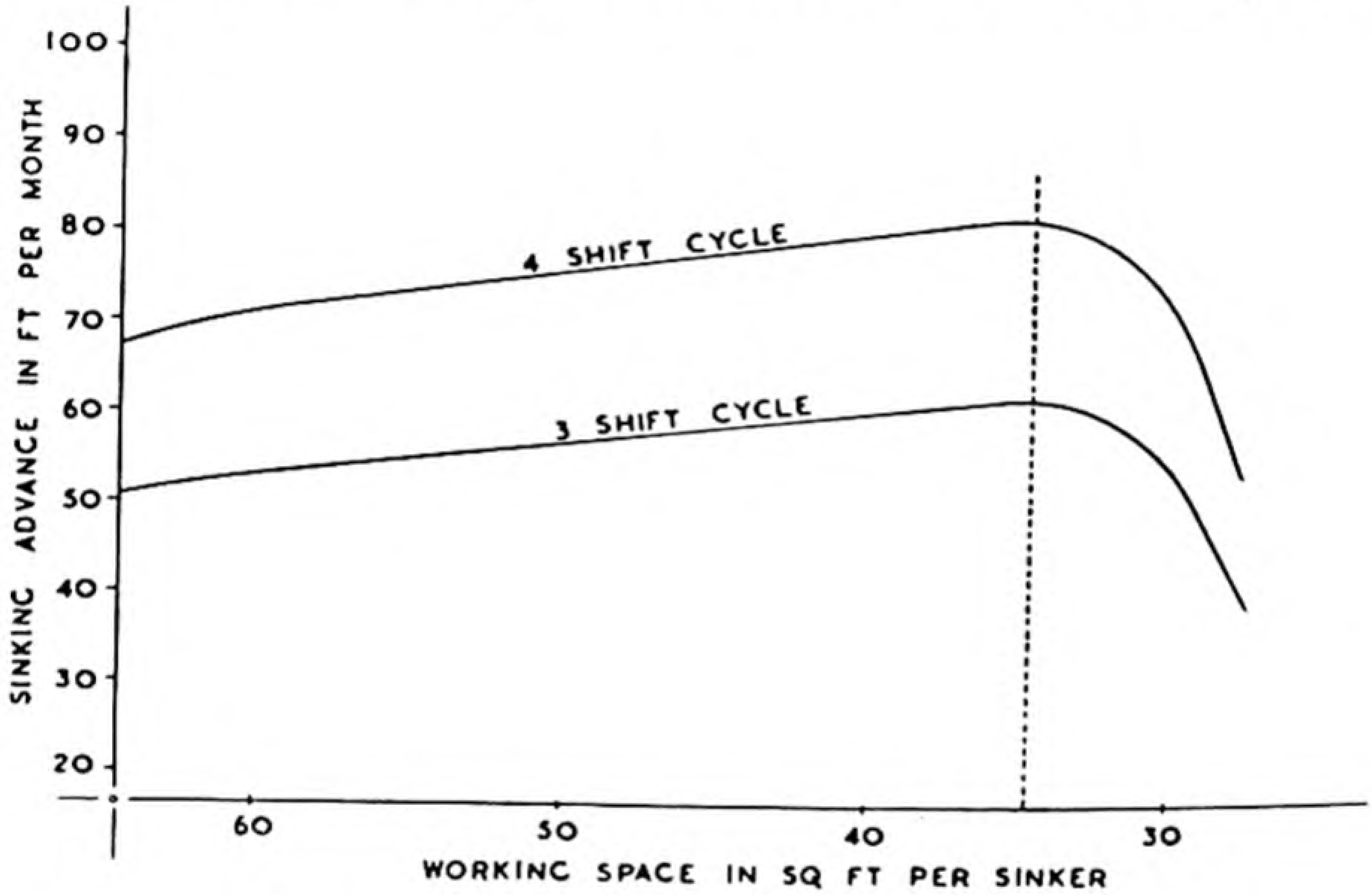


FIG. 186

available working space per man and a high performance per manshift, however, only a relatively small monthly performance rate can be attained. It is necessary, therefore, to consider a lower rate of performance per sinker in order to reach a higher rate of advance in the sinking. The performance per manshift must not decrease below a certain minimum, or, in other words, the number of sinkers employed at the shaft bottom must not exceed a definite optimum. Fig. 186 illustrates how the monthly rate of advance in the sinking depends upon the working space per sinker for teams of three and four men working on a three-shift cycle. Both curves show that the maximum rate of advance is achieved with a working space per sinker of 34 square feet. It follows that with an available shaft area of

130 square feet a team of four men per shift will give the best results. With four-men teams on a three-shift cycle, or twelve men per day, a monthly advance of 20·7 yards can be obtained. With a four-shift cycle and four men per shift, or sixteen men per day, a monthly advance of 27·3 yards can be made. These rates of advance are for sandy-shale. Where the sinking is in shale, the rates must be multiplied by 1·1 to 1·25, and in hard sandstone by 0·8 to 0·9.

In circular staple shafts, with an area of approximately 215 square feet, the maximum advance will be reached with a working space per sinker of 30 square feet. This condition is obtained with a crew of seven men per shift, or twenty-one men per day on a three-shift cycle, or twenty-eight men per day on a four-shift cycle.

The highest sinking performances do not give the lowest costs. The most appropriate sinking cost will be obtained with a smaller team per shift and a larger working space per man. In the cases of the shafts quoted, 130 and 215 square feet, the working space per sinker will be between 37·4 and 42·8 square feet, and 34·2 and 37·4 square feet respectively. The most economical number of sinkers to be employed in the same shaft will be three men per shift on 130 square feet and five or six men per shift on 215 square feet.

The depth of the staple shaft does not influence the sinker performance or the advance per month. With the normal shaft up to 220 yards deep, the increased amount of waste and other material to be transported can be hoisted without loss of time for the sinker crew by using two kibbles. The real sinking costs per yard at different depths differ only slightly, due to the small additional power and ventilation costs at the greater depths. The equipment costs for winding, such as hoist chamber, hoist, kibbles and landings, are more or less independent of the depth between the average limits of 100 and 200 yards in depth. In the case of deeper staple shafts, these costs are distributed over a greater sinking depth, and the total cost per yard decreases with increase in depth.

(*b*) *Examples of the organisation for sinking a staple shaft downwards.* The organisation of work during the sinking of a staple shaft can best be illustrated by two examples:

Case 1. The shaft being sunk is rectangular with an excavation area of 116 square feet, corresponding to a length of 4·6 yards and a width of 2·8 yards. The sinking is through strata composed of sandy-

shale and sandstone. The team is made up of nine sinkers, working on a three shift cycle at piece-work rates. A hoist man, who generally discharges the kibble at the upper level, is present at each shift. Only one kibble is hoisted at a time, while a second kibble is being filled to eliminate waiting time. The shaft supports consist of wooden frames, which are installed up to a distance of 20 feet from the shaft bottom as sinking proceeds. Gliding beams or fixed guides are used for guiding the guide skid and the kibble.

1st Day

Morning Shift	*Afternoon Shift*	*Night Shift*
Drilling and firing the cutters and sumpers separately. Loading out.	Loading out previous rounds and drilling the remaining easers and trimmers.	Drilling the corner holes or trimmers and firing two rounds.
3 men	3 men	3 men

2nd Day

Morning Shift	*Afternoon Shift*	*Night Shift*
Loading out waste from corner holes.	Putting in supports, fitting in lagging and guides.	Putting in ventilation tubes, pipe lines and ladders.
3 men	3 men	3 men

One lift is finished in two days. The round gives an advance of 2 yards and a sinking progress of 1 yard per day. The performance per manshift is therefore 1·3 cubic yards or 3·9 inches.

Case 2. The shaft being sunk is circular with a diameter of 18 feet corresponding to a cross-section of 260 square feet. The sinking is through sandy-shales and sandstone, the daily advance being 1 yard. The shaft is brickwork lined in sections of from 8 to 10 yards as the sinking proceeds. There are twenty-four men employed on four shifts, with six men in each shift. The sinking performance is 1·5 inches per manshift or 32·5 cubic feet per manshift.

1st Day

1st Shift	*2nd Shift*
Preparation for drilling, drilling and firing sump holes. (15 holes, each 3 ft. to 6 ft. 6 in.)	Loading and hoisting waste.
3rd Shift	*4th Shift*
Drilling the round holes. (30 holes, each 6 ft. 6 in.)	Continuation of drilling of round holes and firing round. (10 holes, each 6 ft. 6 in.)

2nd Day

1st Shift	*2nd Shift*
Loading and hoisting waste.	Loading and hoisting waste.
3rd Shift	*4th Shift*
Loading and hoisting waste.	Cleaning up floor and trimming sides loading and hoisting remaining waste.

Section 3. The Driving of Staple Shafts Upwards

(*a*) *General arrangements.* The operations involved in driving staple shafts upwards differ from normal sinking in certain essential features. The drilling must be carried out vertically upwards, the drilling crew working from a platform. All materials and supports must be raised to the face, while special ventilation measures may be required due to the drilling and firing operations. The elimination of the hard work entailed in loading the debris may be offset by the disadvantage of having no means of transport for the men, who normally use a ladderway.

The operations included in driving upwards require a compartment (or way) for the debris, a manway, a compartment for transporting materials and a compartment for ventilation ducts and air-pipes. The division of the shaft into compartments can best be carried out by advancing the shaft supports with the progress of the shaft and dividing the shaft into sections with the cross-beams or spreaders. In the case of rectangular shafts, this method is usual, while with circular shafts, from 10 to 15 feet in diameter, a different

method of driving can be adopted. The compartment arrangement for a rectangular shaft is shown in Fig. 187. The waste compartment at one side is separated from the shaft interior by a strong timber lining. Since the spreaders set across the shaft and forming the waste compartment have to withstand the load of the waste material stored in the compartment, they must be strutted. Only such quantities of waste as are required to maintain a suitable working level for the men at the top are loaded out. The quantity of debris which can be stored in the waste compartment will depend upon the volume of material broken from the solid at the face. The ratio of the amount of waste to be loaded to the rock in the solid, i.e. a volume coefficient of 1·5 to 1·8. With coefficients of 1·5 and 1·8, 70 and 90 per cent. of the debris can be loaded out continuously. It is important to keep the waste compartment filled in order to prevent damage to the supports by falling rock. Larger pieces of rock must be broken for the same reason and to avoid any possibility of jamming in the waste compartment. Complete emptying of the waste compartment is not carried out until the shaft is completed.

Experience has shown that the flow of debris from the waste compartment cannot be carried out successfully if the height is greater than from 85 to 100 yards. Where deeper shafts are being driven upwards, the waste compartment should be interrupted to reduce the pressure caused by the load of waste material, by deflecting the waste compartment to one side. This is done by enlarging the compartment on one side of the shaft, for several yards.

Fig. 187 illustrates the working platform, consisting of strong timber planks and covering the individual compartments, and the firing platform which is built in the same way.

Fig. 187 shows a longitudinal section of a staple shaft in progress, illustrating the air-duct, ladderway, waste compartment, the winch set at the bottom of the shaft for lifting up materials, and the fixing of the rope pulley on the working platform. The alternative arrangement used for the disposal of the waste in a circular shaft at least 15 feet in diameter is shown in Fig. 188. A solid socket of rock supporting the weight of waste material is left in the lower part of the shaft. Another feature of this arrangement is the inner waste chute built into the waste compartment and extended with the progress of sinking. This waste chute is circular, from 20 to 30 inches in diameter, and constructed from steel $\frac{3}{8}$ inch in thickness. The chute is

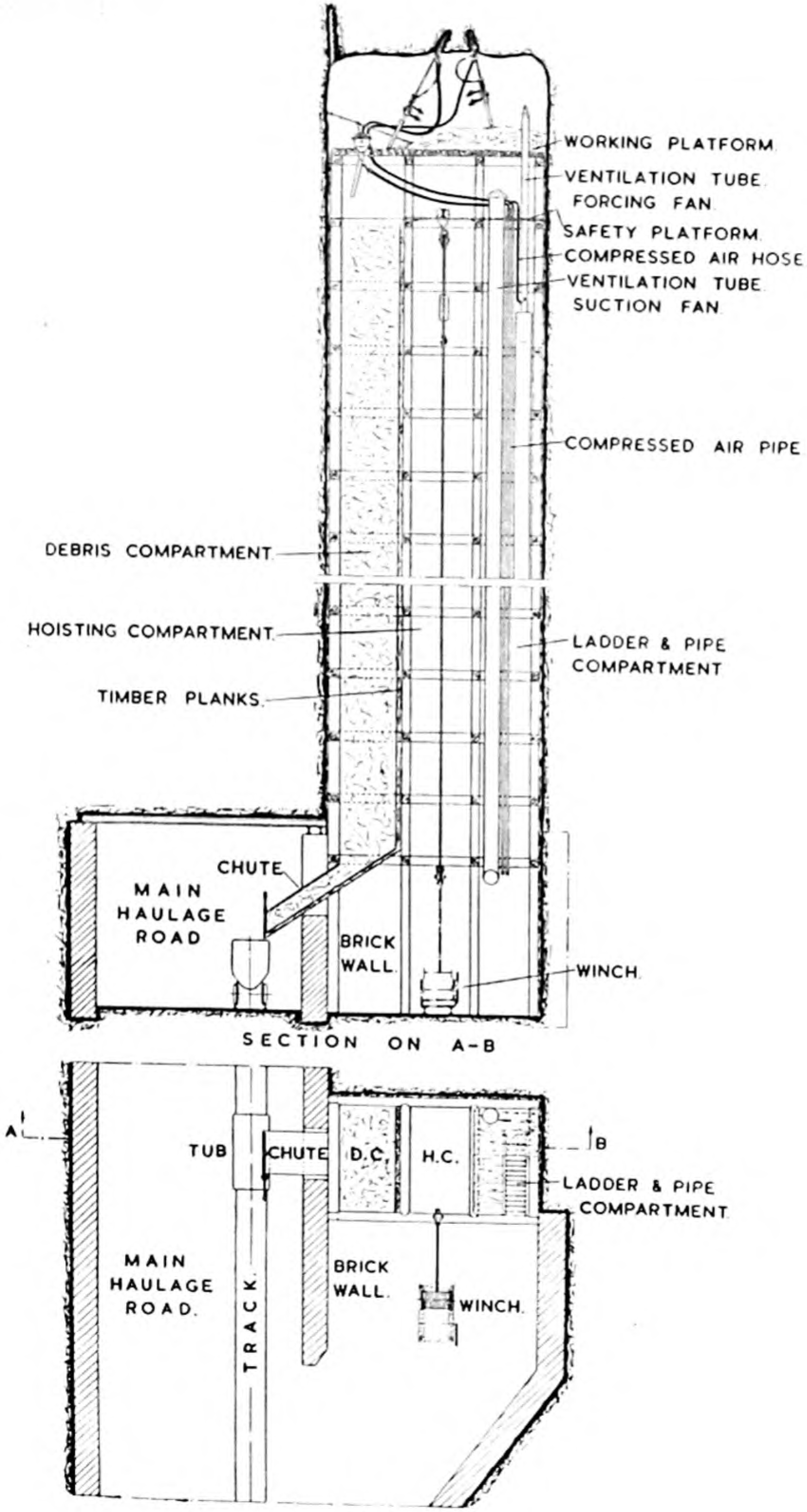

WORKING PLATFORM
VENTILATION TUBE
FORCING FAN
SAFETY PLATFORM
COMPRESSED AIR HOSE
VENTILATION TUBE
SUCTION FAN
COMPRESSED AIR PIPE
DEBRIS COMPARTMENT
HOISTING COMPARTMENT
LADDER & PIPE
COMPARTMENT
TIMBER PLANKS
CHUTE
MAIN
HAULAGE
ROAD
BRICK
WALL
WINCH
SECTION ON A-B
A
B
TUB
CHUTE
D.C.
H.C.
LADDER & PIPE
COMPARTMENT
MAIN
HAULAGE
ROAD
TRACK
BRICK
WALL
WINCH

Fig. 187

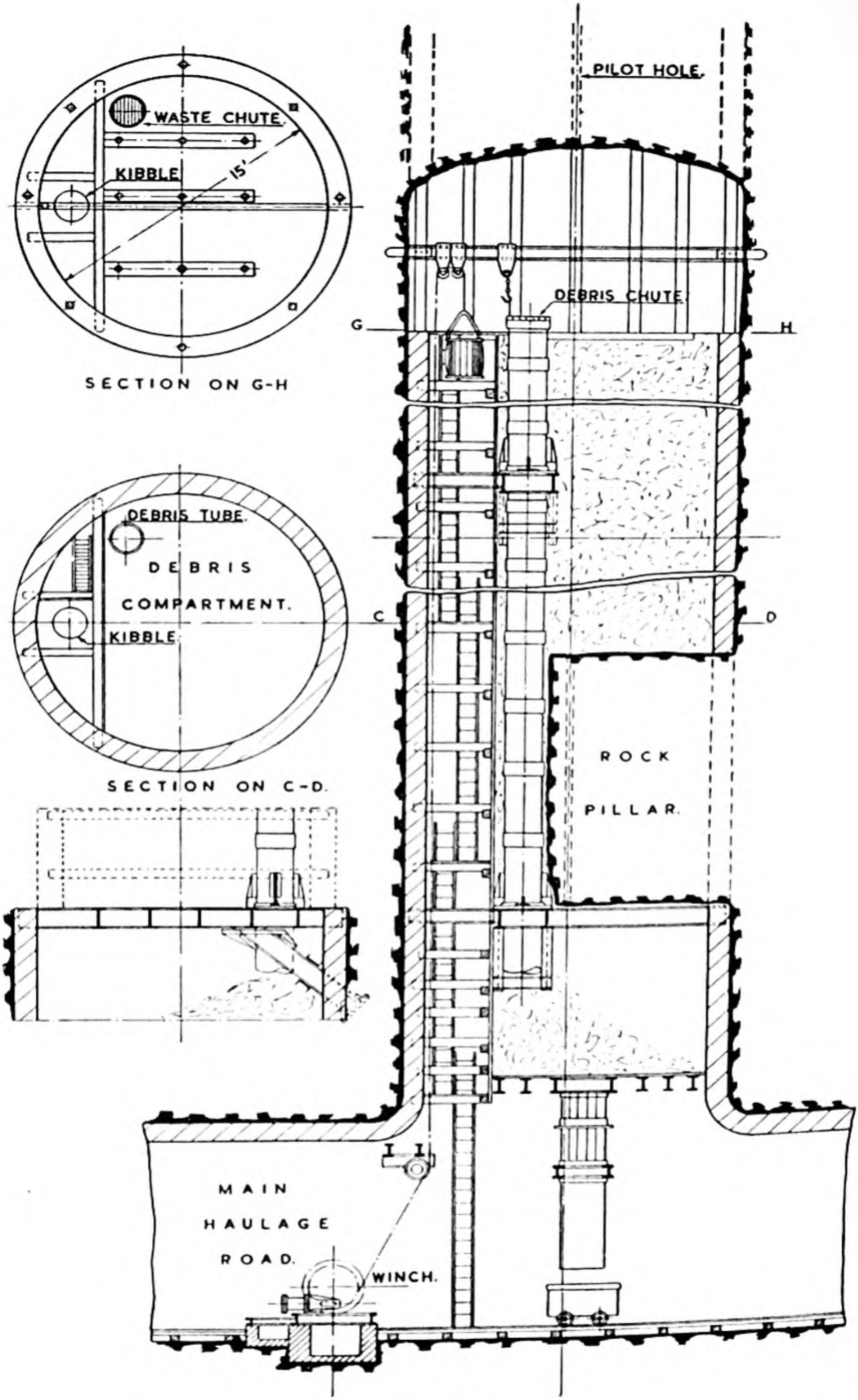

WASTE CHUTE
KIBBLE
15'
SECTION ON G-H
DEBRIS TUBE
DEBRIS
COMPARTMENT.
KIBBLE
SECTION ON C-D.
PILOT HOLE.
DEBRIS CHUTE.
G
H
C
D
ROCK
PILLAR.
MAIN
HAULAGE
ROAD.
WINCH.

FIG. 188

covered by a grid at the top and serves for the disposal of surplus waste which cannot be stored in the waste compartment.

The adoption of the spiral chute as a waste compartment during the raising of a staple shaft is frequently carried out, but only where

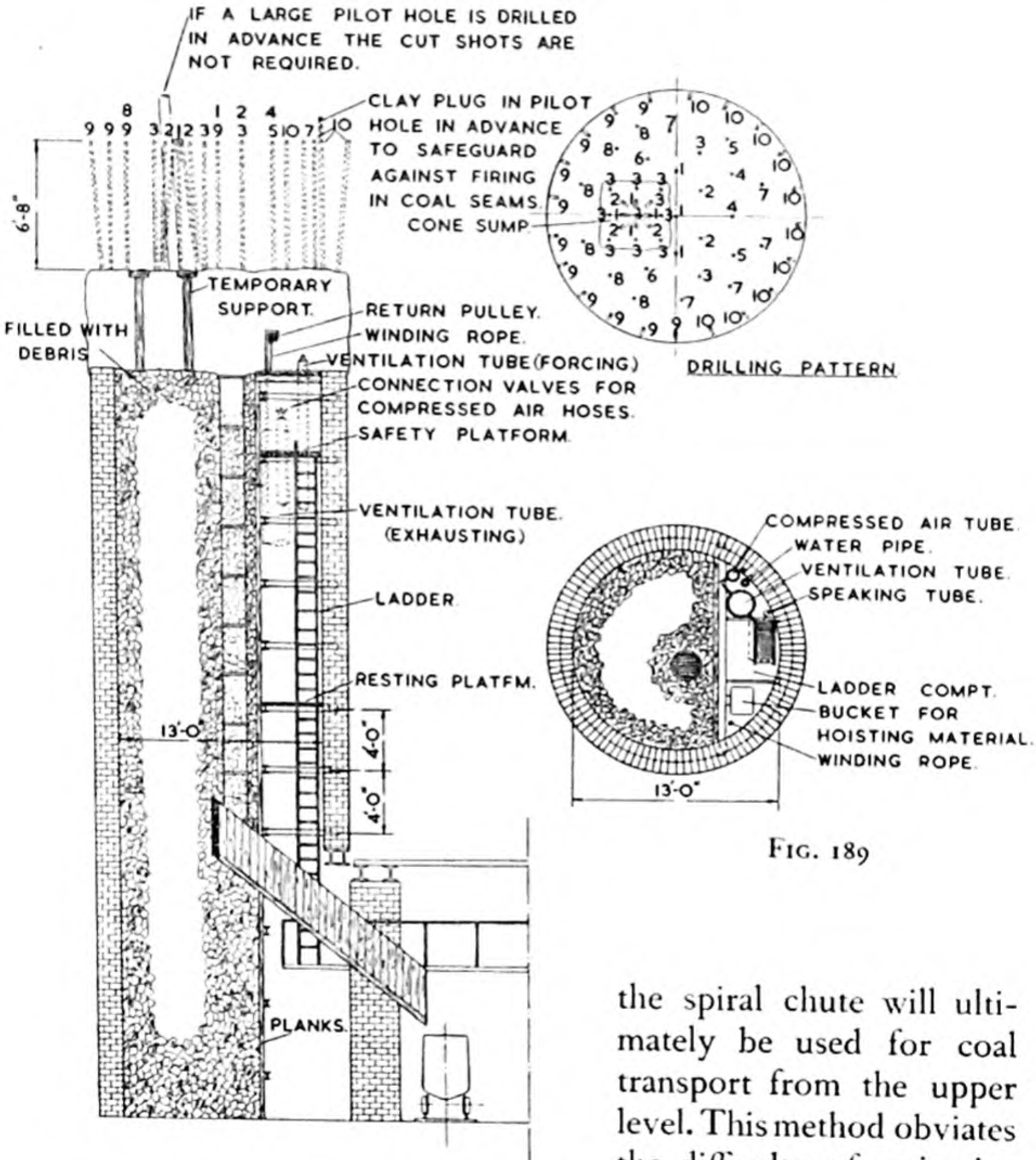


FIG. 189

the spiral chute will ultimately be used for coal transport from the upper level. This method obviates the difficulty of maintaining the waste compartment. When the spiral chute has a cast basalt lining, it is inserted after the shaft has been finished, the lining being too brittle to support the waste to be fed into the chute.

Driving a shaft 15 to 18 feet in diameter upwards and taking the full cross-section as illustrated in Fig. 189 is not to be recommended unless the rock is strong and there is no tendency for the shaft sides

to cave. It may also be necessary to support the shaft sides with wooden cribs and punch props.

Another method for driving large section staple shafts, which can be applied under all strata conditions, is to drive a small pilot shaft (10 × 8 feet or 12 × 8 feet) upwards first. The pilot shaft is subsequently enlarged, from the top to the bottom, to the finished shaft size. The stripping of the sides of the pilot shaft is carried out with explosives, the pilot shaft compartments being used for waste, men, materials and air and ventilation lines. The permanent supports of the enlarged shaft are set from the bottom to the top, consisting of a timber lining or lagging kept in position by cribs set 6 feet apart, as described in Chapter 3, Part III, Section 4.

The ventilation arrangements for driving operations require special attention, since accumulations of methane or noxious gases must be avoided. In the case of deep mines, the air temperature also is important. In such cases, a connection to the upper level, giving 'through' ventilation, can be made by driving a borehole, from 10 to 18 inches in diameter, from the lower intake to the upper return level. Another advantage of this exploratory hole is the reduction in the number of shot holes required per round. The cost and time required to drill this hole will be partially or wholly balanced during the sinking of the shaft. Generally, these boreholes are drilled from the lower level due to the ease in the removal of the chippings from the hole. The drilling is done either by percussive or rotary drilling machines. Where a heavy hammer drill is used, the bit is the single chisel type. The hammer drill used for this work is designed to operate with pneumatic feed. The drilling machine, illustrated in Fig. 190, is mounted between two compressed air-operated columns and is attached to movable cylinders which operate on the fixed piston rods or columns so that the feed can be regulated. This feeder device can be used also when attaching and detaching drill rods. The drilling speed depends upon the nature of the ground; with sandy shale it will be from 6 to 12 feet per shift, and with sandstone from 3 to 5 feet. In the last few years, special rotary drilling machines, manufactured by the German firms Nuesse & Graefer (Sprockhoevel) and Wallram (Essen), are increasingly used for this purpose. With these machines the quoted drilling speeds are more than doubled.

The deviation of the borehole from the vertical should be kept as

small as possible and, even if the borehole does not run exactly on the shaft centre, it should not finish beyond the boundary of the shaft.

Where the strata is flat, deviation of the hole should not present a problem. Where the strata is inclined, it is advisable to commence the hole inclined towards the general inclination of the strata in order to reduce the probable deviation. With holes from 100 to 140 yards long, it should be possible to keep the hole within a shaft area of 130 square feet, but for greater lengths exploratory boring becomes more difficult.

Where drilling is being carried out from the top level, deviation can be kept to a minimum by using free-fall drilling machines which impart a short, quick blow to the bit. This type of equipment can seldom be used, since it requires a derrick at least 12 feet high and a mud flush system for the removal of the borings. It is probably twice as expensive as drilling from the lower level.

A further means of improving the ventilation, where this is required, is to use a dual air-duct system, incorporating two fans, one forcing and one exhausting.

Wet drilling becomes disagreeable when drilling upwards, and other means of removing the dust produced during drilling have been

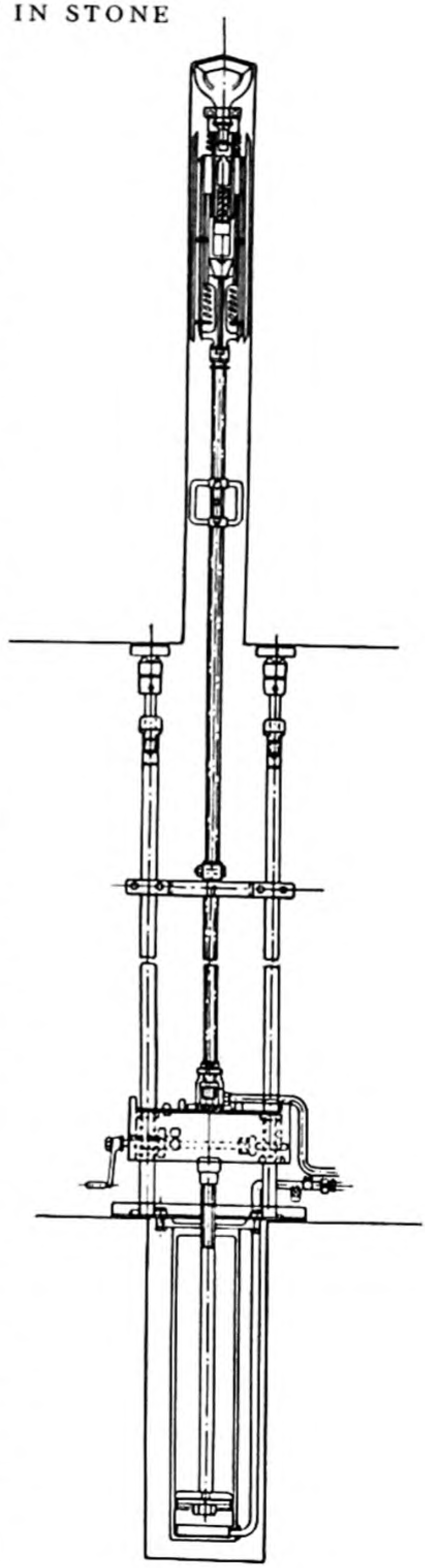


Fig. 190

tried. Removal by suction devices and the use of sprays for dust suppression have been successful.

(*b*) *Sinking performance and costs.* Where a shaft is being sunk downwards, the sinker performance, all other conditions being

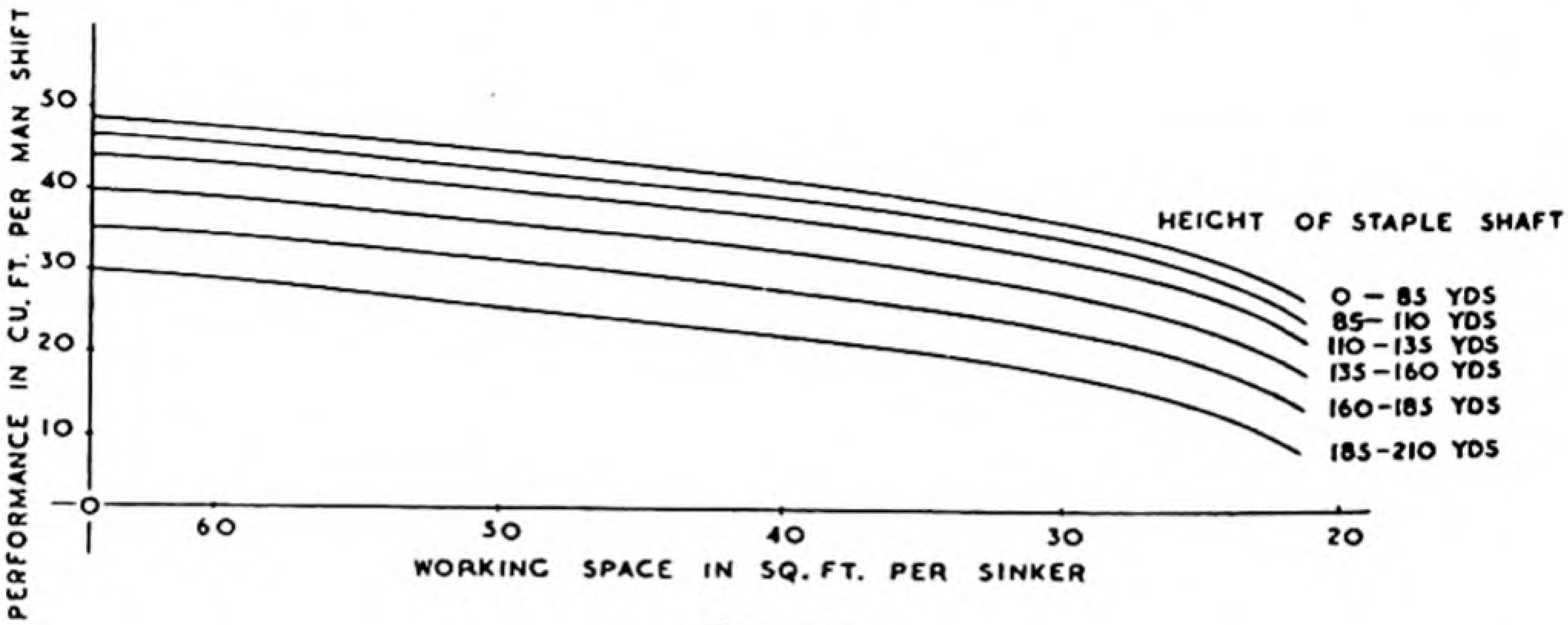


Fig. 191

equal, is dependent upon the working space per man. In driving upwards, the performance is also dependent upon the height, since the men must climb ladders to the face, and the transport of materials is more difficult. The ventilation and waste disposal also will become an increasing factor with greater height. The influence of these factors on the performance will become more apparent above a height of approximately 80 yards, causing a reduction in the sinker performance as illustrated in the following table:

Height of Shaft	*Sinker Performance*
yards	Per cent.
80	100
80–110	95
110–140	90
140–165	80
165–195	70
195–220	55

On the whole, the performance per manshift will be about 20 per cent. higher during driving upwards than when sinking, due to the deletion of loading work. For the same reason, the performance will not be reduced so much with decrease in working space per man, as shown in Fig 191, which shows the relationship between sinker performance, working space and height of shaft. If the performance is

1·82 cubic yards at a working space of 65 square feet, it will still be 1·04 cubic yards when the working space is reduced to 21·5 square feet. The monthly progress increases with decreasing working space per man, and reaches the best performance of 44 yards when the height of the shaft is 80 yards and the working space 23·5 square feet, *vide* Fig. 192. With greater height, the best performance is reduced and is about 19·6 yards for a height of between 80 and 110 yards. The most favourable working space increases from 23·5 to 32 square feet. Similar to ordinary sinking, the lowest cost per yard is not within the range of the highest driving performance, but at performances corresponding to a working space of from 38 to 48 square feet per man.

(*c*) *Example of the organisation for driving a staple shaft upwards.* The staple shaft to be driven is a 14 by 9 feet rectangular shaft. The sinking crew consists of eight men working in four shifts, with two men in each shift. There is, in addition, one man at the shaft bottom who operates the winch and loads out waste from the hopper. The individual operations are as follows;

1st Day

1st Shift	*2nd Shift*	*3rd Shift*	*4th Shift*
Drilling.	Drilling.	Drilling the remaining holes, firing the sump inner and outer holes.	Stripping the face and loading the waste into the waste compartment. Cleaning up the firing platform.

2nd Day

Firing the trimmer holes, stripping the face and cleaning platform.	Cleaning continued, loading waste from hopper and transporting waste.	Setting support frames, lining the waste chute advancing air pipe-lines.	Installation of drilling platform ready for drilling.

A lift of 6·5 feet is finished in two working days, giving a drivage of 3·25 feet per day. The performance per manshift is 1·69 cubic yards, or 4·9 inches.

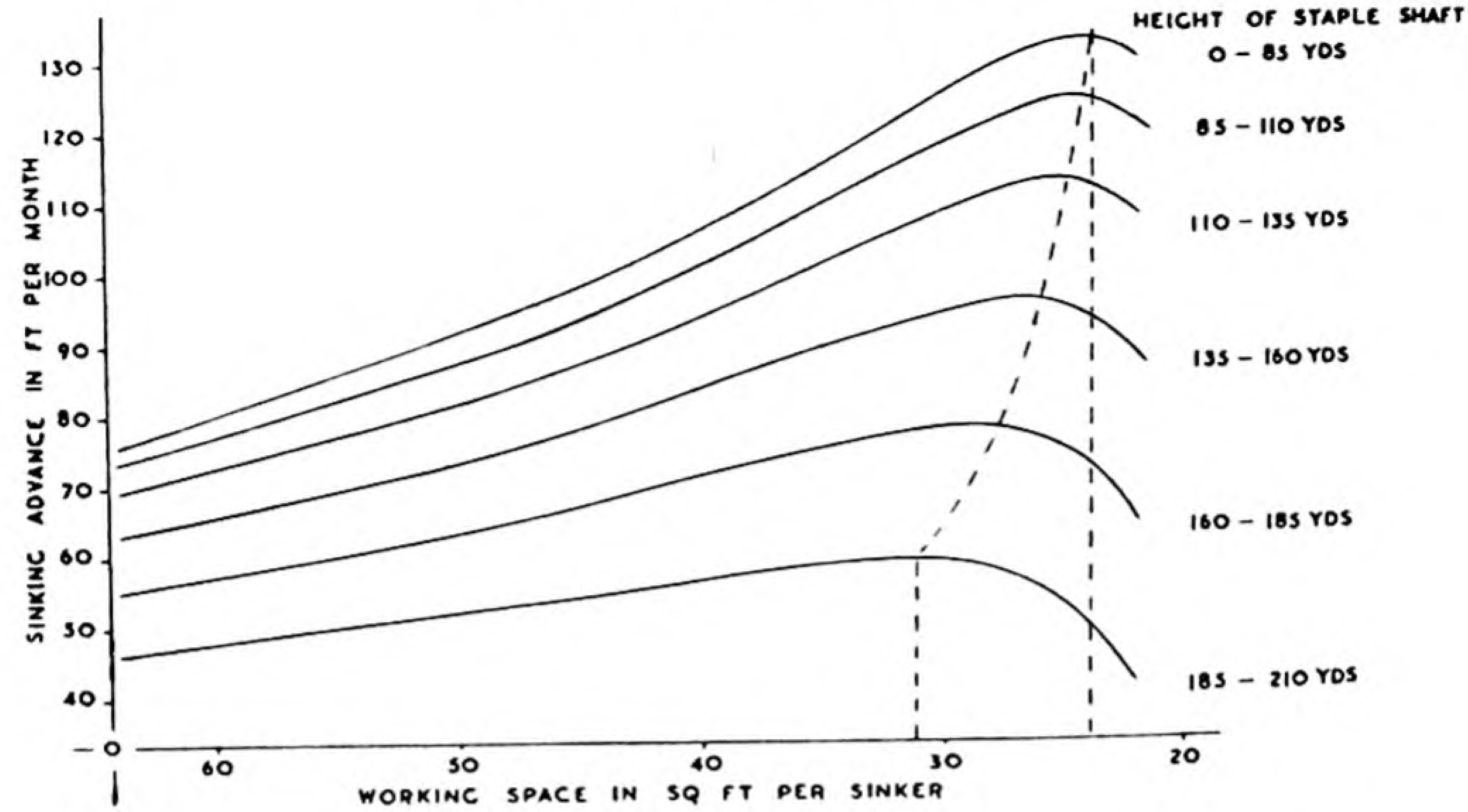


Fig. 192

(*d*) *Factors to be considered in deciding on normal sinking or driving upwards.* In addition to the time and cost involved in driving a staple shaft of a certain depth, other factors must be considered in deciding in which way the shaft is to be driven.

Considering the time factor, it has been found that the average monthly progress when driving upwards in shafts up to 550 feet is slightly higher than that for sinking, excluding the time required for the pilot borehole. The total cost for a shaft of 300 feet which is driven upwards will be somewhat less than for normal sinking, the difference being about 10 per cent. After a depth of 375 feet, there is little difference in the total cost. Consideration must be given to shafts sunk to depths less than 300 feet. The following factors must be considered:

1. The particular type of work requires skilled stonemen, although a certain proportion of unskilled men can be used.

2. In the case of deeper shafts, driven upwards, the drilling of the pilot borehole becomes more difficult.

3. Wet drilling in driving upwards is disagreeable.

4. When the waste compartment is more than 250 feet high, stoppages are likely to occur, hindering the sequence of operations.

Consideration of these factors favours normal sinking, but the additional cost and drainage difficulties which may be encountered

in sinking downward should be remembered. The storage or disposal of the waste from the sinking will depend upon the availability of cheap transport for the waste at either the upper or lower levels. If the transport is cheaper via the upper level, sinking will be preferred, and vice versa. It may be the case that the lower level is in course of development and sinking must be carried out from the existing upper level.

This question arises only where shafts are being sunk between two levels. Where the staple shaft finishes between levels, the direction of sinking will depend upon the method and direction of development, which may be under-level extraction or from the lower level upwards.

Section 4. The Support of Staple Shafts

(*a*) *Rectangular staple-shaft supports.* Rectangular shafts are usually supported with timber and seldom with steel. Timber which has been treated with preservatives is preferable on account of cost

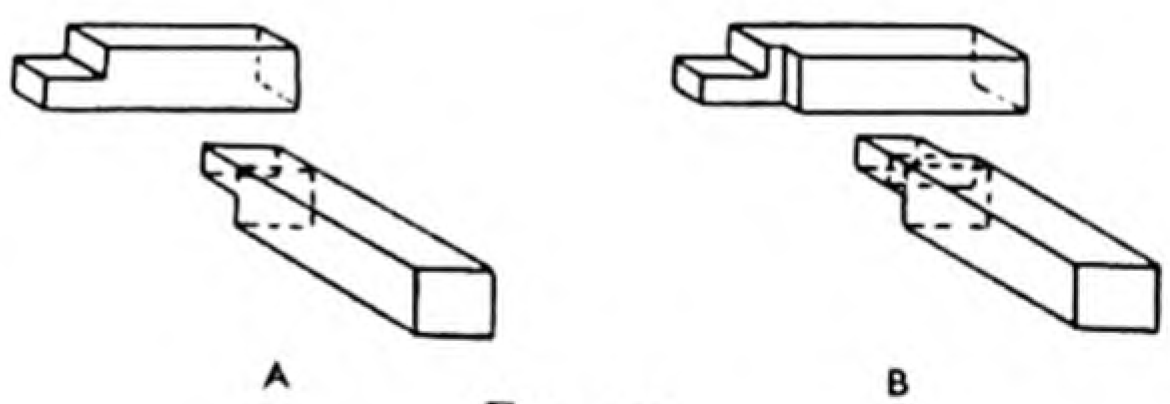


FIG. 193

and ease of repair, the latter being an important factor when considering the effect of extraction on the stability of the shaft.

In the case of important staple shafts oak is preferable due to its toughness and strength, though treated pine is also used. The main section of the supports is the rectangular frame. To enable the frame or crib to withstand both vertical and lateral pressure, the bars are connected by scarf joints. The simplest scarfing is illustrated in Fig. 193(A) in which the thickness of the tenon and shoulder cut is arranged to correspond to the direction of the maximum lateral pressure. Where there is a possibility of oblique pressure, the frame can be constructed as in Fig. 193(B), which illustrates a compound scarf jointing in which both vertical and horizontal scarf joints are used.

Generally, the timber is square, and round timber is seldom used

for this purpose. The timber dimensions increase as the shaft section increases. Under ordinary conditions, with shafts varying in cross-sectional area from 75 to 115 square feet, side bars and end bars 7 by 7 inches are usual.

In many cases, the timber sets are inserted 3 feet apart and cross-braced by bolts from each corner. Where the strata pressure is high, the distance between sets may be reduced to about 1 foot; they are seldom laid 'skin to skin' as in close-joint timbering. In order to take the weight of the timber sets, it is recommended that bearer bolts should be notched into the shaft sides every 15 to 30 feet. The shaft sides are closely lagged between the frames with strong oak or pine planks. Where a shaft is being driven upwards, the supports are continually advanced to within 3 or 6 feet from the face. During normal

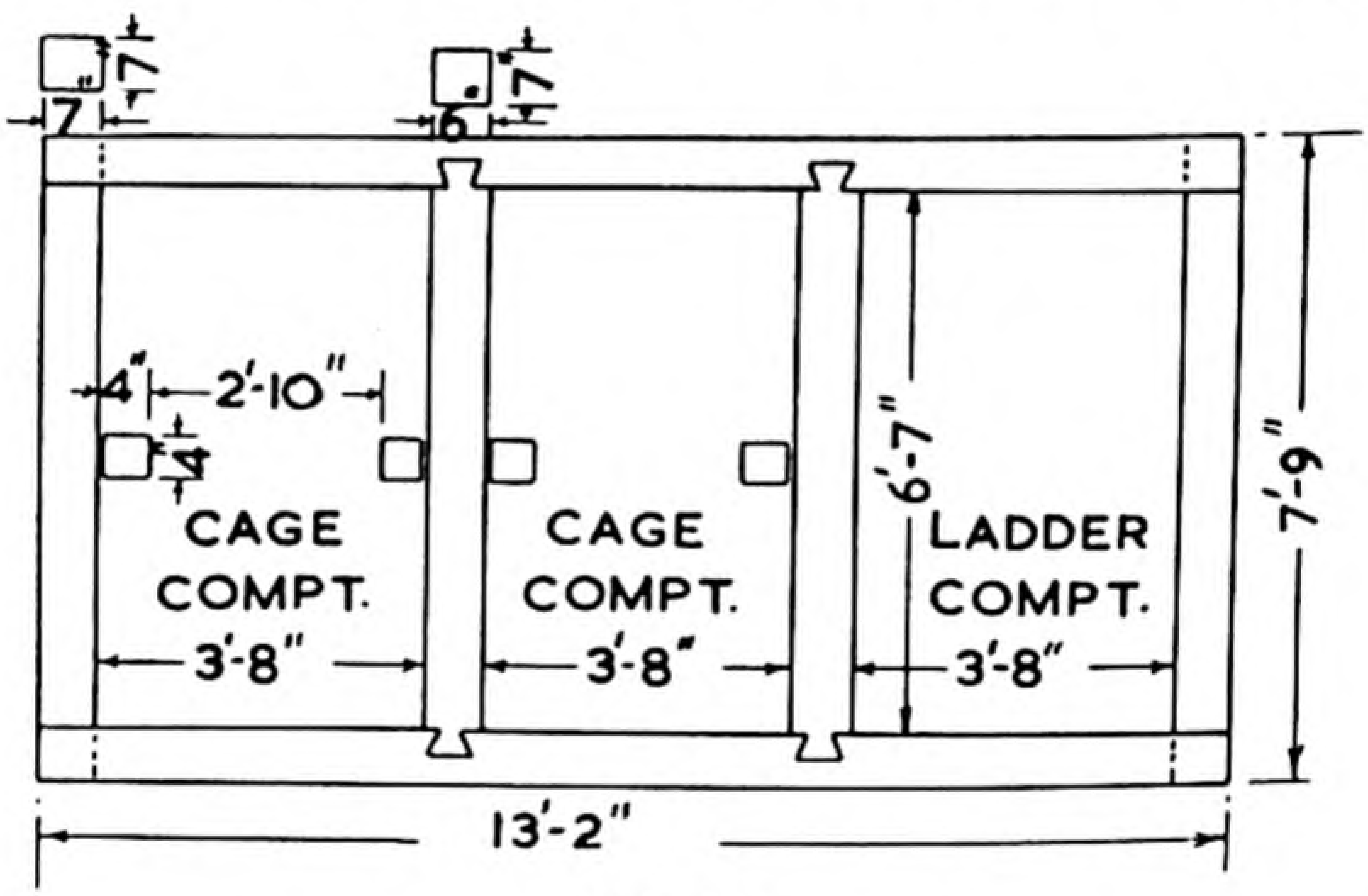


FIG. 194

sinking, two methods may be used. Where the strata is fairly solid, sinking and insertion of supports are carried out alternately at intervals of from 30 to 150 feet; where the strata is less solid, the interval is reduced to from 15 to 30 feet. In order to afford maximum support to the shaft sides, the supports will be inserted up to 6 feet from the shaft bottom. The supports are set, therefore, from the top downwards, following the sinking progress, and the frames are suspended from the previous frames by steel hangers. The use of bearer bolts notched into the shaft sides is again recommended.

The sub-division of the shaft is carried out using cross-struts or buntons. These partition bars also serve to strut the long side bars,

reinforcing the support. These bars must be connected firmly to the side bars and are usually dovetailed, or a short tenon is set into a gain on the end-bar, as shown in Fig. 194.

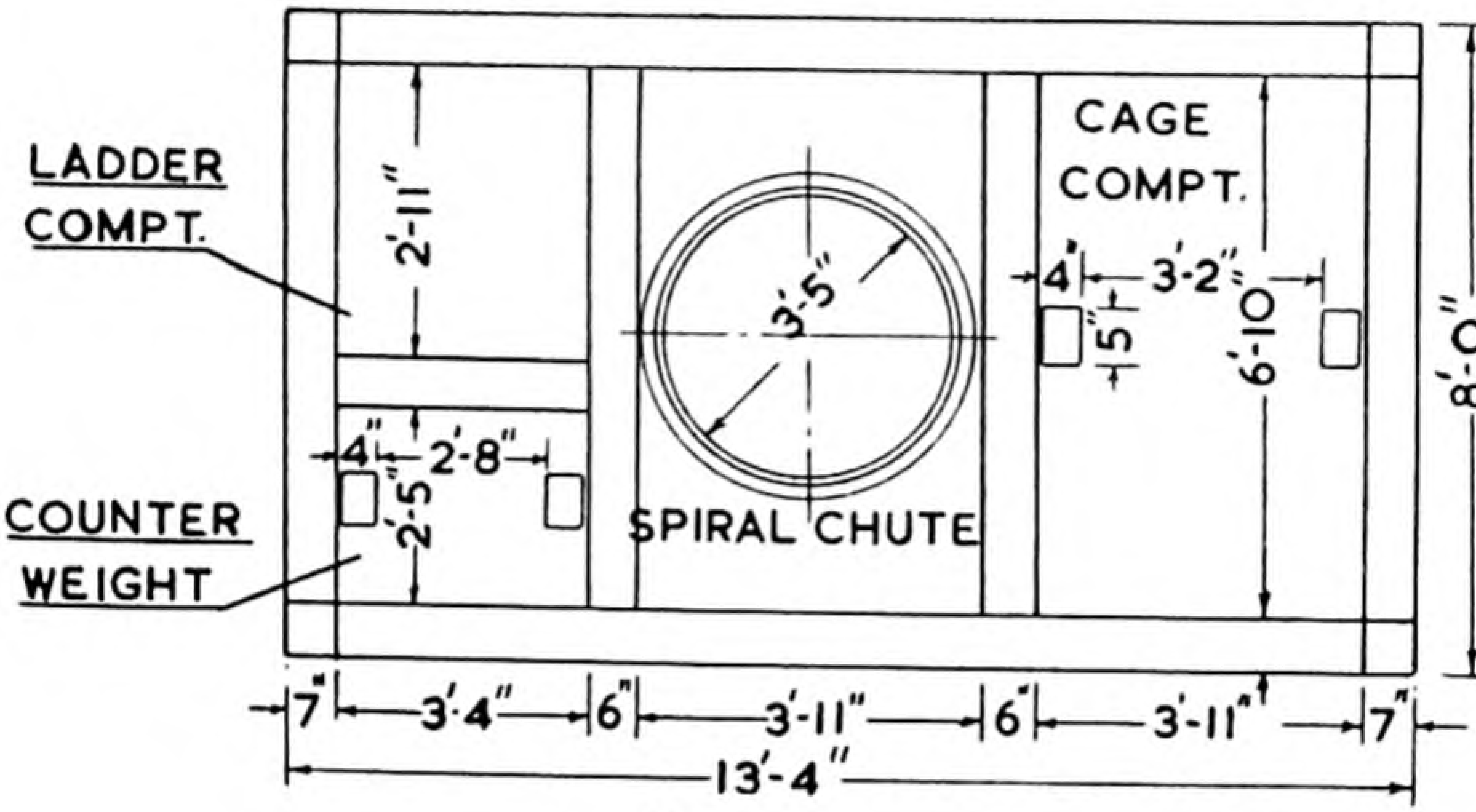


FIG. 195A

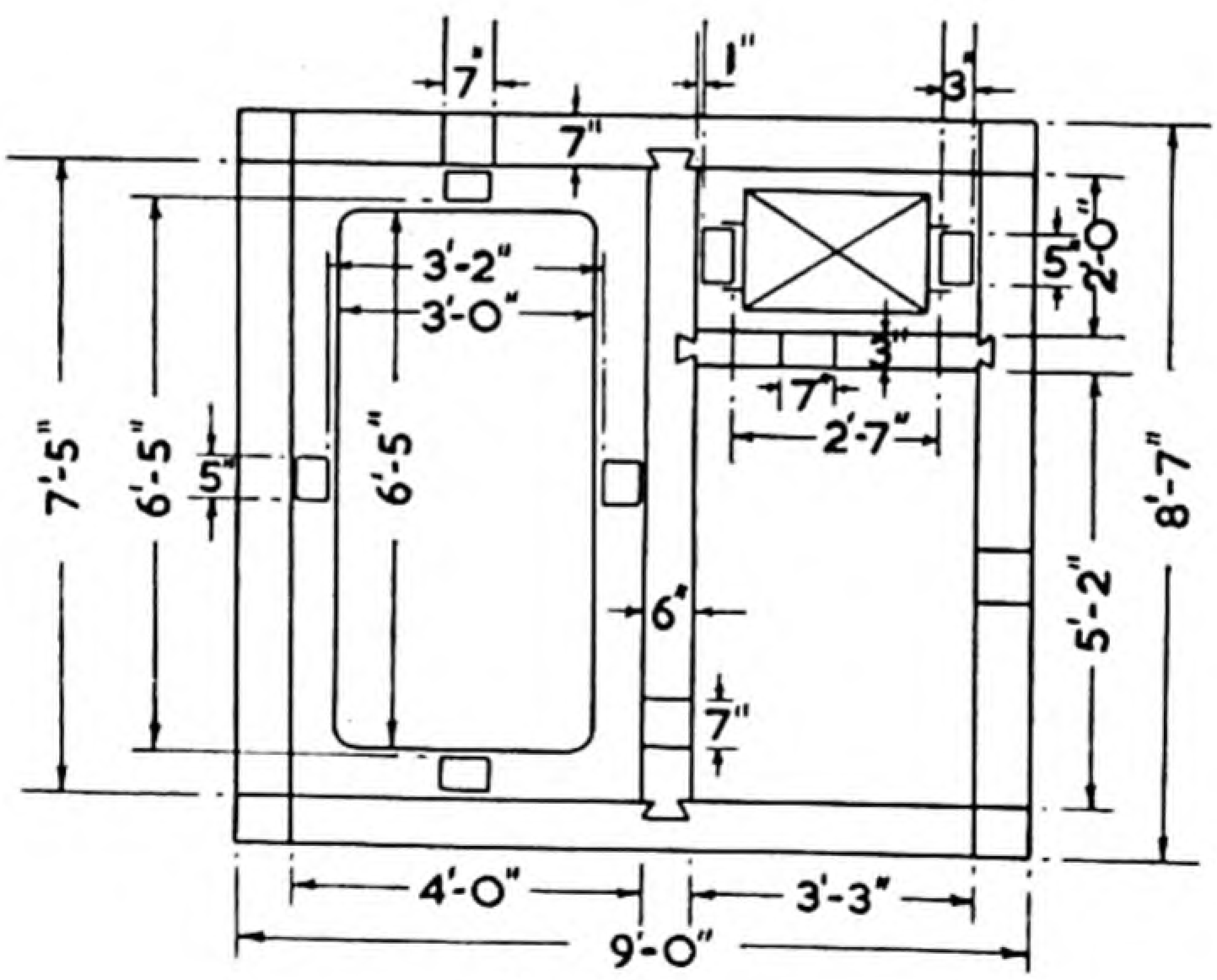


FIG. 195B

The staple shaft shown in the section in Fig. 194 is equipped to take two cages, while Fig. 195 illustrates a single-cage winding shaft with and without a spiral chute installation. Where a rectangular

shaft is being supported with steel frames, the cribs are usually 'I' section, channels, or a double-channel section in which single channels have been riveted back to back. The side- and end-bars are joined at the corners by cutting the girders diagonally at the ends and bolting them together with angles, or as illustrated in Fig. 196, in which the double-channel sections are scarfed and bolted together.

As in the case of timber cribs or frames, the distance between sets is determined by the strata pressure conditions. In order to transfer the weight of the support to the strata, frames with projecting sections are used at distances of from 15 to 30 feet apart, the ends being notched into the strata. Alternatively, short steel bearer sections can be grouted into the shaft sides, bearing on every fifth or tenth crib. It is also necessary to strut between the crib sections with strong round timber and lag behind. The lagging may be of wood or of steel sheeting. The spreaders or partition bars can be fastened easily and securely to angles bolted on to the main frame. Fig. 197 shows details of this form of support, and the section illustrates the layout of the permanent steel structure.

(*b*) *Circular staple-shaft supports.* Circular staple shafts are usually brick-lined, but steel-ring supports are used occasionally. The brick lining is generally one and a half or two bricks thick, the bricks being laid by cross or header jointing, the former method being stronger due to the uniform distribution of the joints. Cement-mortar should be used, consisting of one part of cement and three parts of sand. The cavities behind the brickwork should be well filled in order to strengthen the lining. Drainage pipes, $1\frac{1}{4}$ inches diameter, should be inserted through the brickwork to prevent any possibility of water building up behind the walling while the mortar is setting. After the mortar has set, these pipes are closed with blind flanges or wooden plugs. The pipes can also be used for cementation of the strata behind the walling and for this purpose the pipes are usually threaded at the end to take a stop valve, *vide* Fig. 198. Where the shaft is being driven upwards, the brickwork is built up from the bottom, following the shaft as it proceeds. As bricking is being built up in stages of from 30 to 50 yards, similar sections of the shaft have to be temporarily supported according to the nature of the strata. The temporary support consists of steel rings or channel-section cribs, built up in segments. The segments are usually butt-

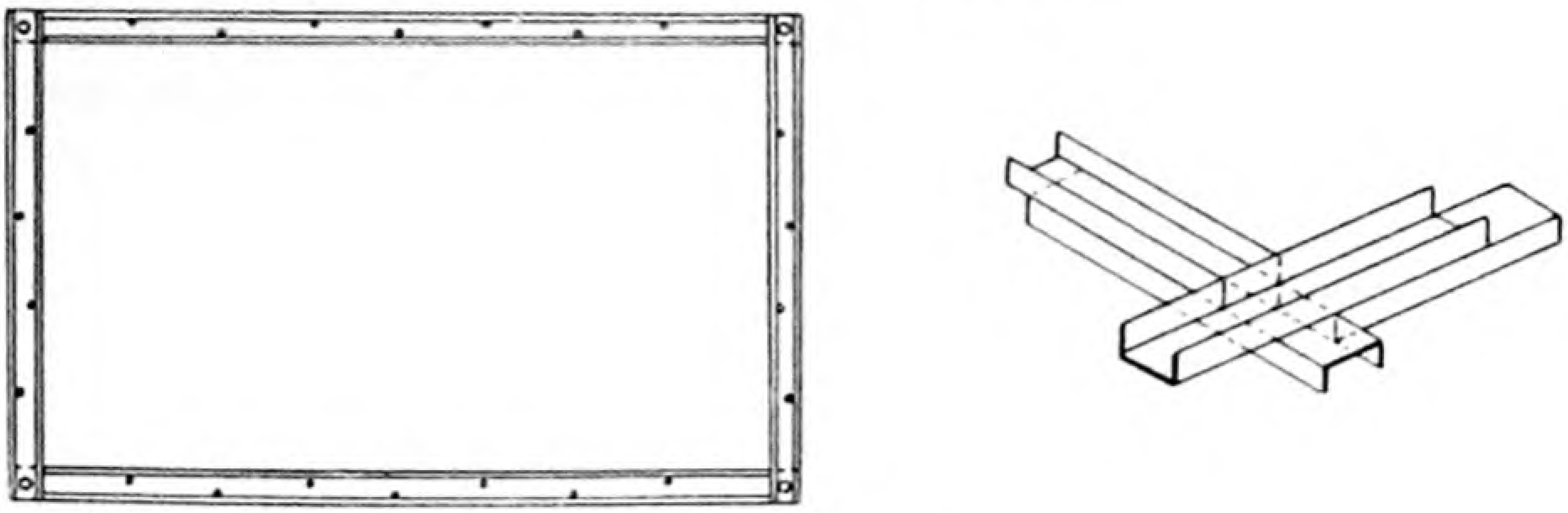


FIG. 196

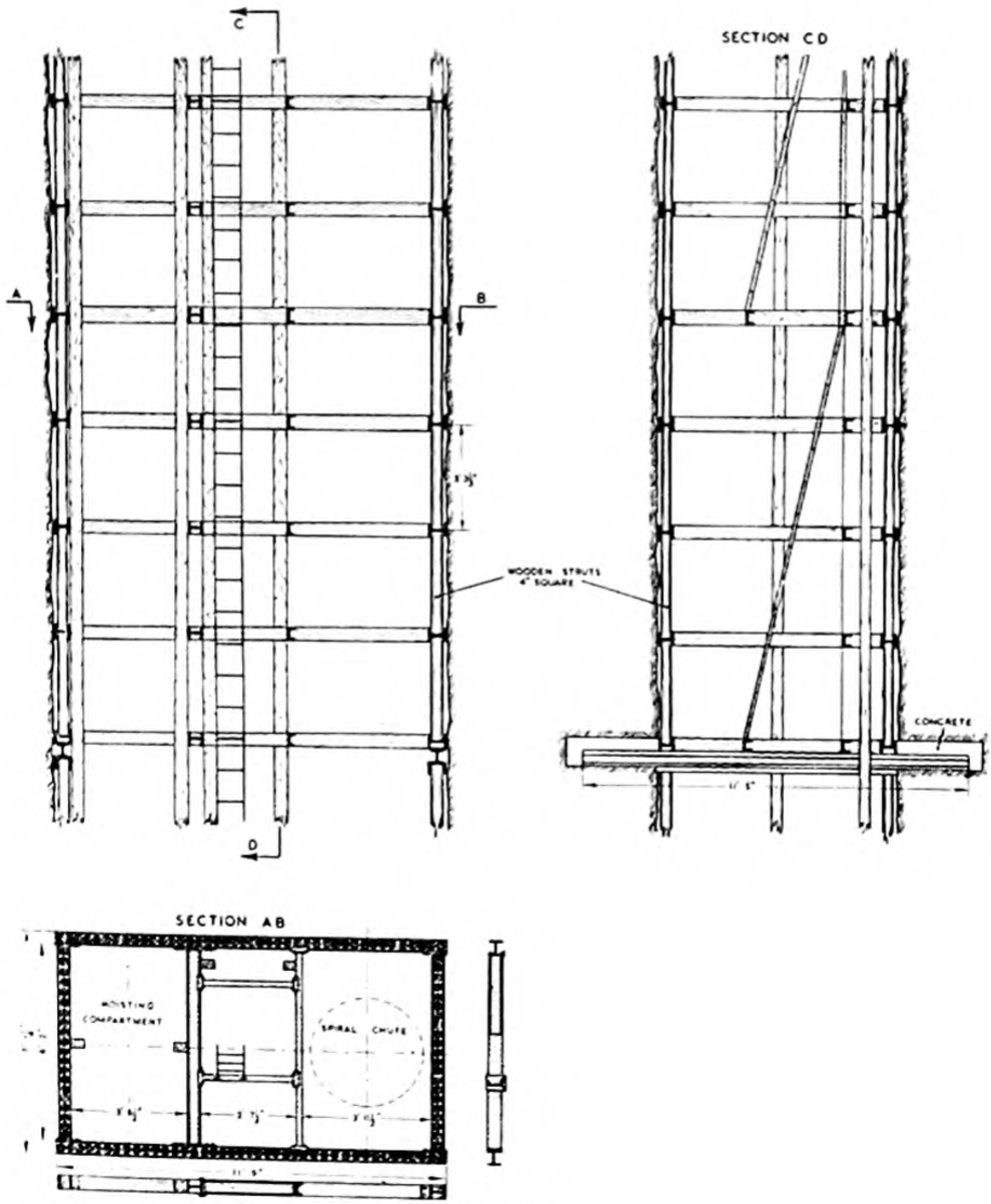


FIG. 197

jointed, with overlapping fish-plates. The first ring is fixed on to a special girder framework at the upper level or to hooks which have been embedded into the finished brickwork. The succeeding cribs are suspended from each other by loose hangers as shown in Fig. 199. It is recommended that every fifth or sixth ring should be supported by horizontal bolts set into the shaft side. Steel sheets, $\frac{1}{8}$ inch thick, or planks are used for lagging behind the cribs. The steel sheets are

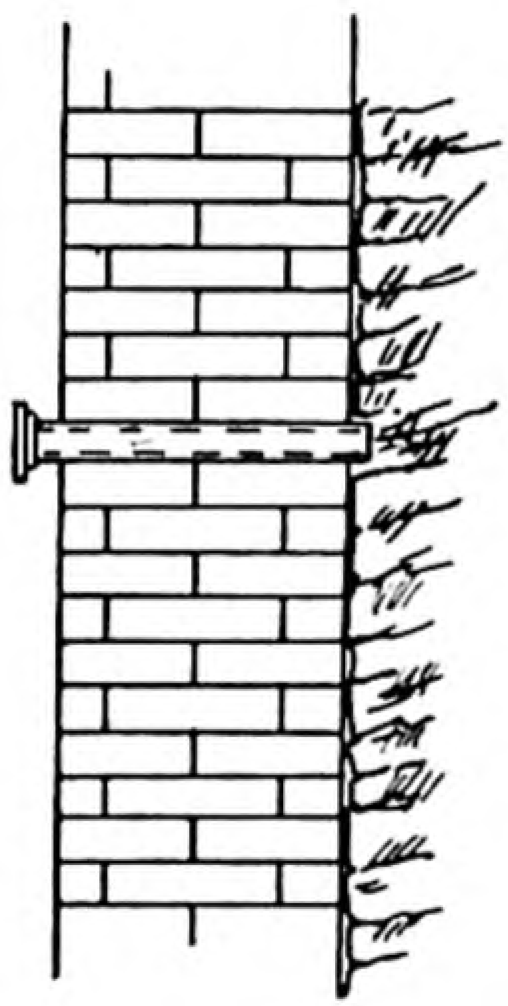


FIG. 198

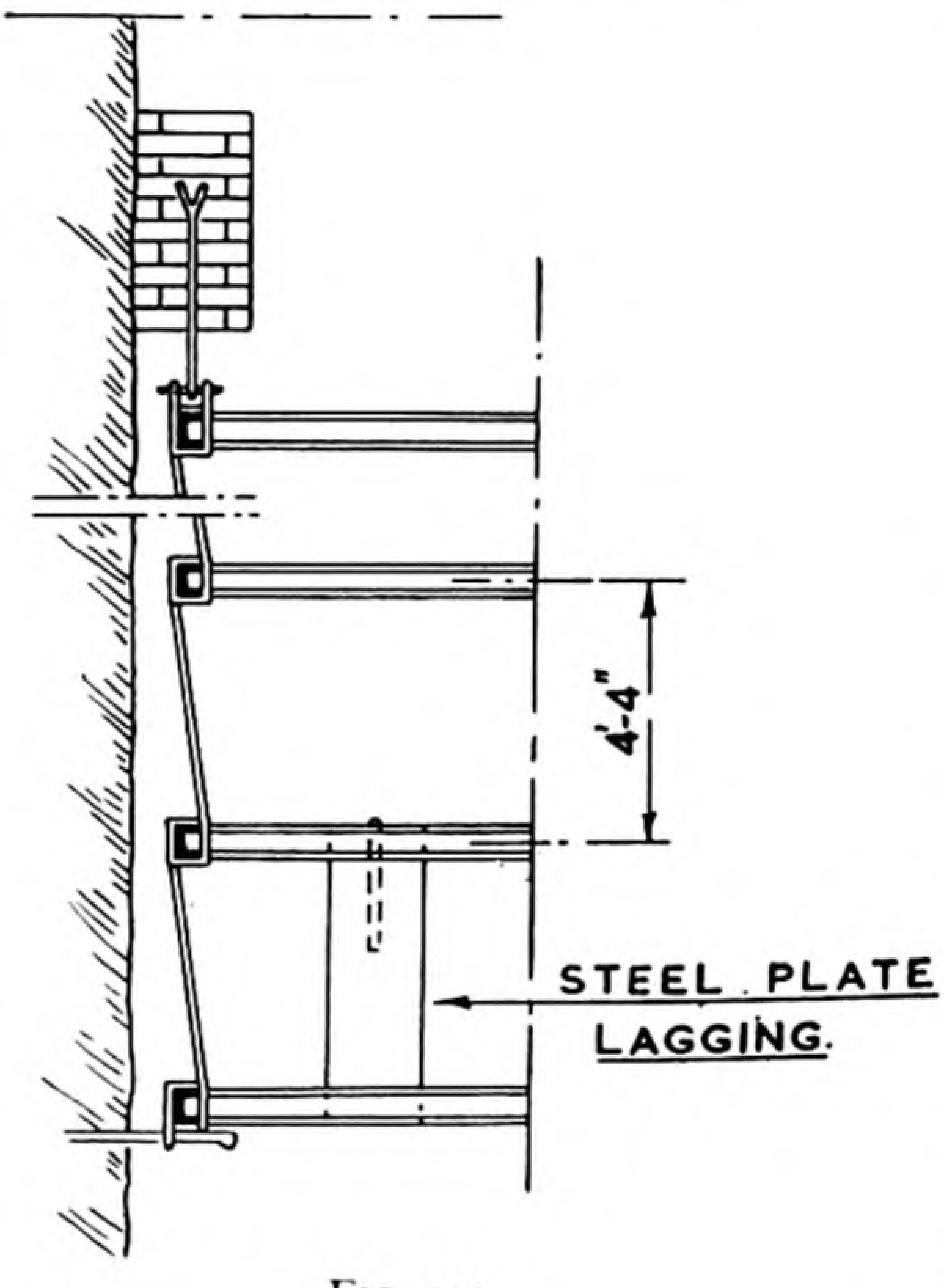


FIG. 199

FIG. 200

provided with suspension hooks, the planks being wedged tight to the shaft sides with wedges driven between the cribs and the lagging or supported on the lower ring by little blocks nailed in the planks. The rings and lagging can be used repeatedly, *vide* Fig. 200.

A bricking platform or walling scaffold is provided from which to insert the brick lining. The scaffold consists of a cradle fixed to a wire rope attached to a winch at the upper level so that it can be raised and lowered as desired. The scaffold is round and fits within the finished diameter of the shaft. It consists of a girder framework

of 'I' or channel sections, covered with strong planks. As shown in Fig. 201, the scaffold is provided with openings to take the ventilation duct and compressed-air main to the shaft bottom. The scaffold

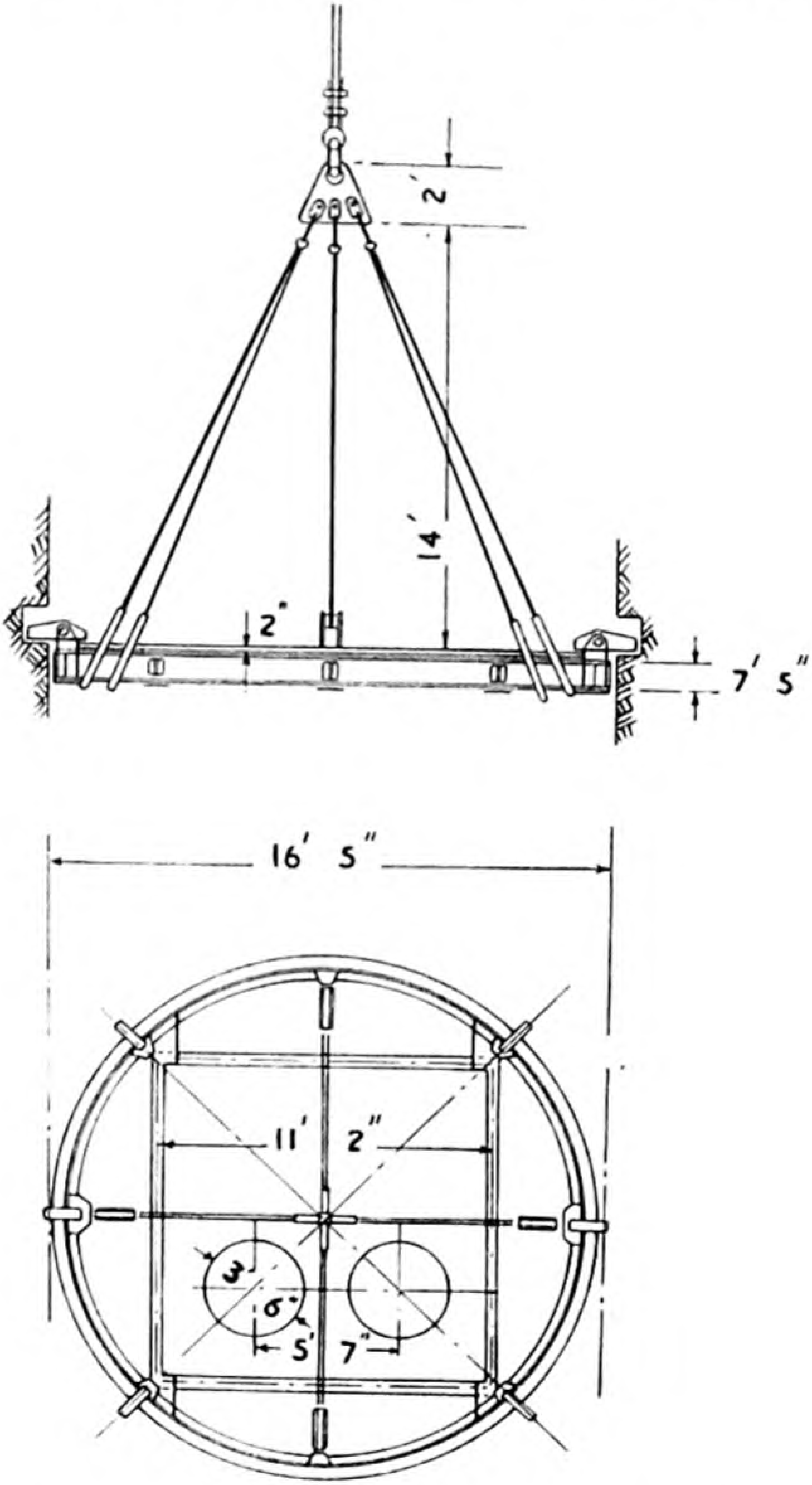


Fig. 201

has also six flap bolts which can be used to suspend the scaffold from the shaft walling as an additional precaution. When the cradle is being lifted, the bolts can be raised to a vertical position. This form of supporting bolt is better than the simple slide bolt, which may

cause the platform to tilt if not fully retracted before lifting.

Each section of bricking is commenced from a walling crib set into the strata. The walling crib bears the succeeding brickwork until it is set, after which the brickwork, which has been closely connected to the shaft side, is self-supporting.

Two forms of walling crib are shown in Fig. 202(A) and 202(B). The form shown in A can be used in less solid strata, since the side pressure is transferred obliquely outwards. The brick walling is continued from the inset crib as shown. In the case of the form

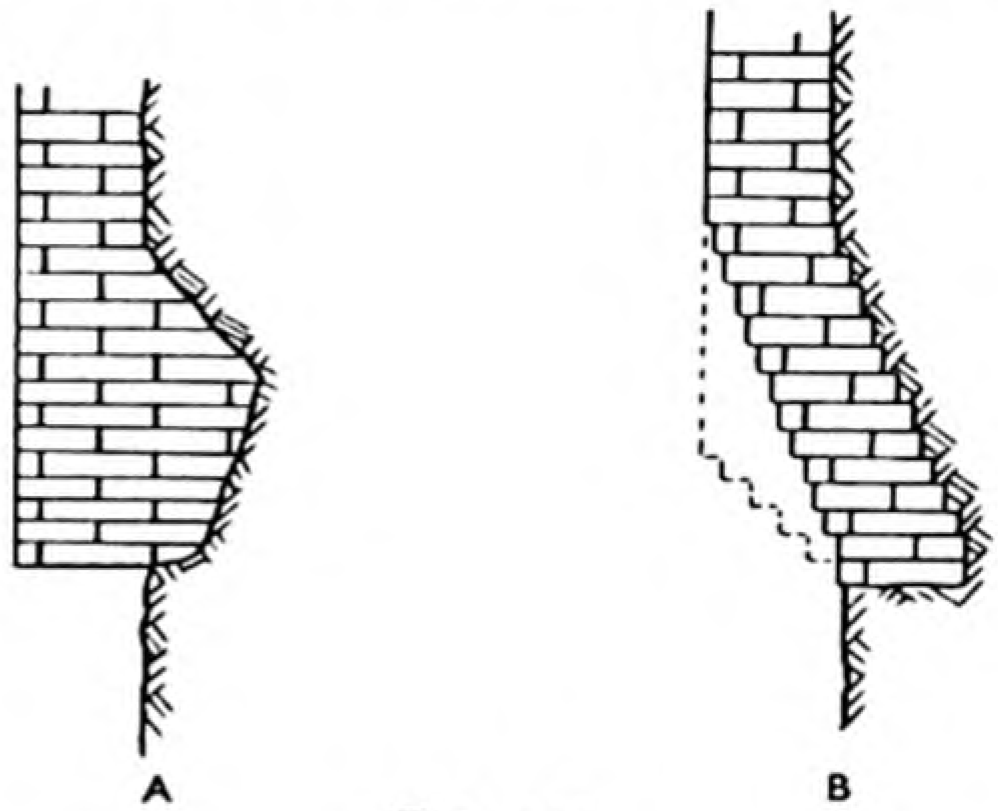


FIG. 202

shown in B, the wide formation is omitted by stepping the brickwork out to the finished shaft size. The stepping out to the finished size can be completed usually in eight courses where a double-brick lining is being inserted.

Another method of walling is shown in Fig. 203. The bricking up is carried out in short stages of from $1\frac{1}{2}$ to $2\frac{1}{2}$ yards. The bricking up follows closely behind the shaft sinking and is built up on steel rings. The rings consist of steel channel sections provided with a steel sheet covering which is as wide as the finished brickwork. The rings are fixed to rods suspended from hooks embedded in the previous brickwork section, the length of these rods depending upon the length of the bricking stage. After sinking a further loft, the ring is lowered into position using chains fastened to the spring hook on the hoisting rope. The ring is then fastened to the lower side of the previous brickwork, using the embedded hangers, and the brickwork started again. The brickwork cannot extend to the shaft bottom, but follows at a height of at least 2 yards to prevent it being damaged by blasting. A simple way of providing a working plat-

form is by leaving the debris as a working stage until the walling is completed, after which it is loaded out. This method is mainly used where the strata is less solid. With stronger strata, this method also has the advantage of dispensing with the brickwork scaffolds which are necessary when longer brickwork stages are used. The spreader timbers can be inserted within a short distance of the shaft bottom, simplifying the guiding of the skip or hoppit, and the fixing of ventilation and compressed-air lines.

The buntons for the division of the shaft into the individual com-

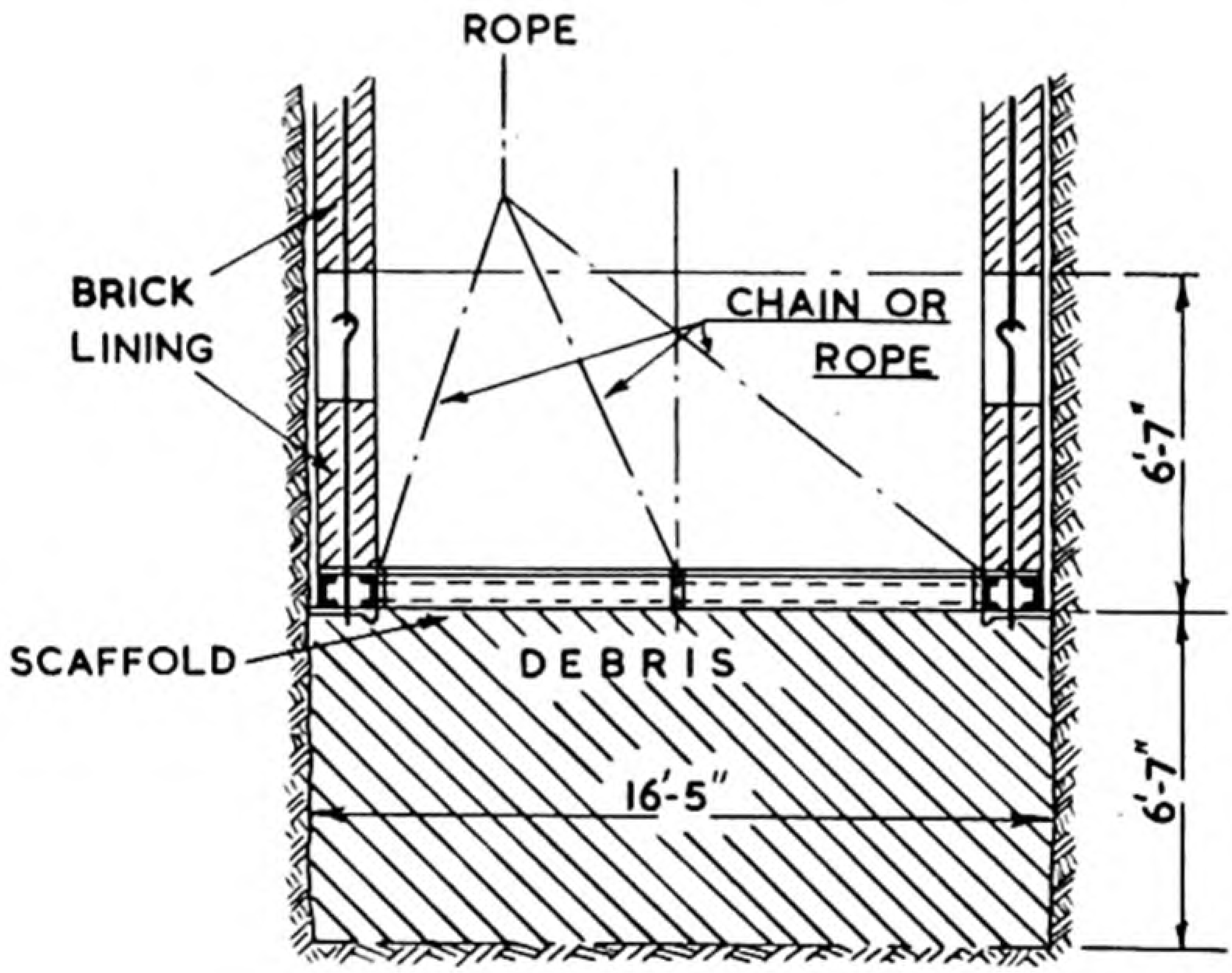


Fig. 203

partments and on which the gliding beams or guides are fixed may be either timber or steel sections. The buntons can be inserted only while driving the shaft upwards. In sinking, the buntons are inserted in openings in the brickwork after the bricking has been completed.

Simple steel ring supports are used only for staple shafts which are sunk in solid strata, since this form of support cannot withstand heavy strata pressure. This form of support is similar to the normal temporary ring support, except that the channel segments are connected by bolted butt joints, the ends of the segments being boxed. The individual rings are connected by vertical steel channel sections bolted to the flanges of the individual rings, as shown in Fig. 204.

The steel sections are well lagged with oak lagging wedged behind the cribs and vertical supports. As in the case of temporary supports, every fifth or sixth ring should be supported by steel rods

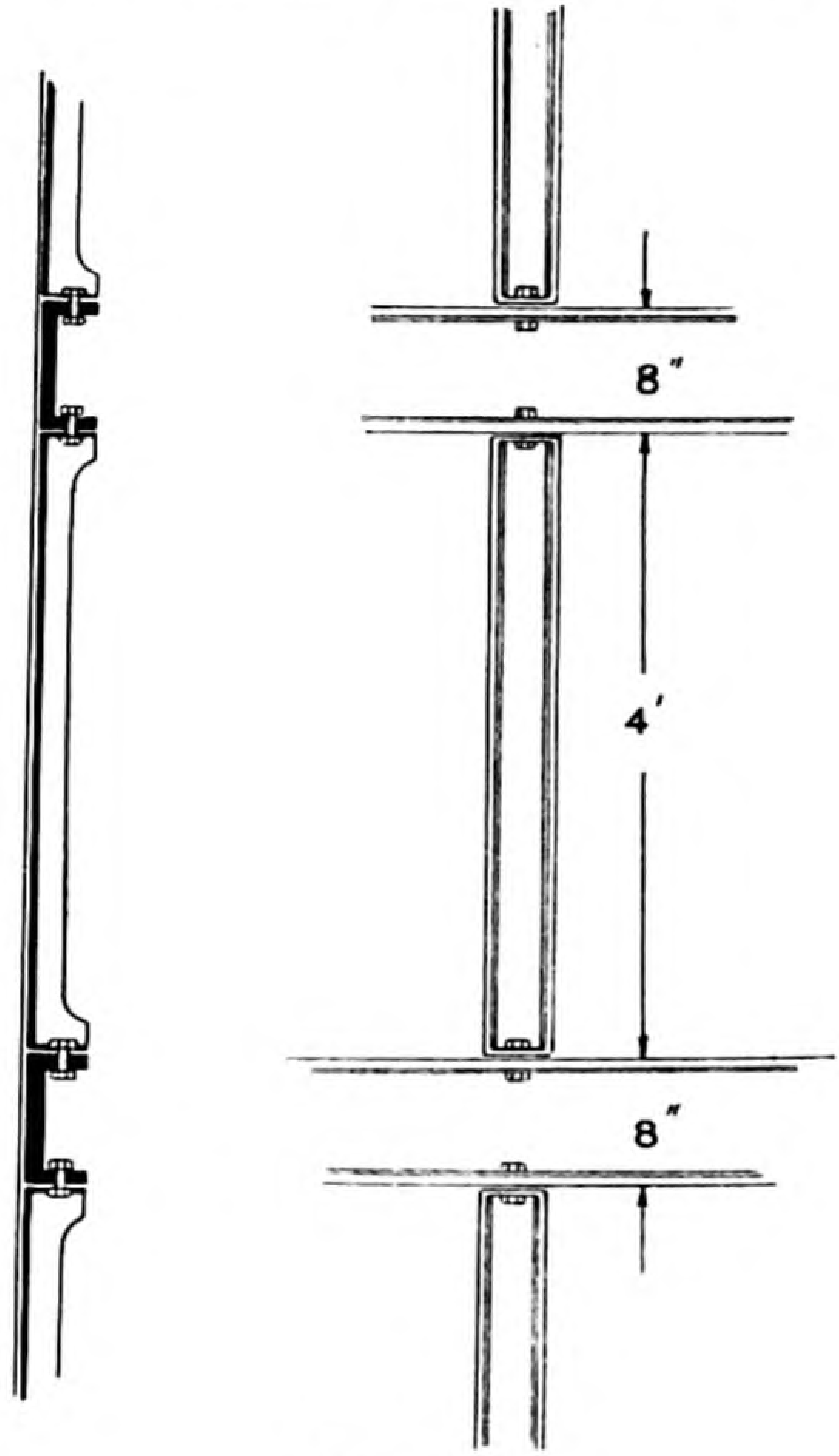


Fig. 204

inserted into the strata. The rings can be positioned by wedging the crib into the correct position.

In the case of steel ring permanent supports, steel or timber buntons can be used. Steel buntons can be secured with angle bars to the cribs, while timber buntons can be fastened using shoes attached to the steel cribs.

CHAPTER 4
DEVELOPMENT IN COAL

PART I

THE DEVELOPMENT OF GATE ROADS

Section 1. *General Introduction*

THE gate roads which are driven in the seam have to serve for ventilation, for coal, waste and material haulage, for the transport of machinery and as travelling roads. Where the system of working in flat seams is longwall, either single- or double-unit faces are developed. The single unit is developed between two gate roads, one serving for coal transport and as an intake airway, while the other is used for the transport of materials and waste and as the return airway from the face. In the case of double-unit development there are three gate roads, two tailgates and a centregate. The mothergate, or centregate, serves for coal transport and as an intake or return, and, if pneumatic stowage is being used, the blast pipe-line may be installed in this road. The gate roads on each side can be either intakes or returns, and serve as material and waste haulage roads.

On the other hand, in flat measures, gate roads can be also maingates or trunkgates serving the secondary gate roads described, the trunk or maingates providing the haulage and ventilation for several faces within a district. In steep measures, an upper and lower gate road are essential, and limit the face length on each side. The lower gate road serves for coal transport and as intake airway, while the upper gate road takes the return air from the face and is used for waste and materials transport.

Where the gate roads are driven in a fairly level seam, they are usually straight and haulage can be carried out with belt conveyors. The straight direction of the roads means that they cannot follow major changes in the inclination of the seam; thus, where the gradient changes frequently, curves cannot be avoided; in these cases it is endeavoured to keep the roads horizontal and car haulage is generally adopted.

The gate roads are driven either before extraction takes place or

simultaneously with face development, depending upon the method of extraction being adopted. Where retreating methods are being used, the gate roads are driven before extraction, but, with advancing faces, the roads are driven with the face. These gates can be 10 or 20 or more yards in advance of the coal face or may be ripped just following the advancing face. Thus, the types of gate roads which may be driven can be distinguished as follows:

1. Gate roads developed independently of the coal face and the time of starting coal extraction.

2. Gate roads ripped up to the face and following the advance simultaneously with the coal face.

3. Gate roads which are driven simultaneously with the face advance but which are from 10 to 20 or more yards in advance of the actual face.

The method of development of the gate roads, both in the coal and the stone ripped to make the road, is the same in all three cases. The second group implies that the coal in the gate is won out simultaneously with the face. There are, however, greater differences in the methods of coal and waste transport in these gates and in the organisation of the development work.

Section 2. Coal Winning

A distinction is made between winning out in the coal and in the adjacent stone in the gate road. It is necessary to ensure separate coal and waste haulage so that the coal is kept as free as possible from ripping material. The coal can be won more easily if the seam within the gate road is removed first, and this ensures a good loose end or sump for the ripping in the adjacent rock.

Sometimes the coal is under-cut, using Shortwall or arcwall coal cutters, the former being preferable. The cut is usually at the floor, but a middle cut also may be used. Vertical shear cuts are unusual, and in any case would only be adopted in seams over 5 feet in thickness.

In most cases the coal is won by shot firing, triangular sumping being mostly practised. If the coal is undercut, sumping is not required, the holes being placed in such a way that, when blasting, the undercut acts as a loose end.

The number of shot holes required depends on the hardness and thickness of the coal seam and the width of the road, but is usually

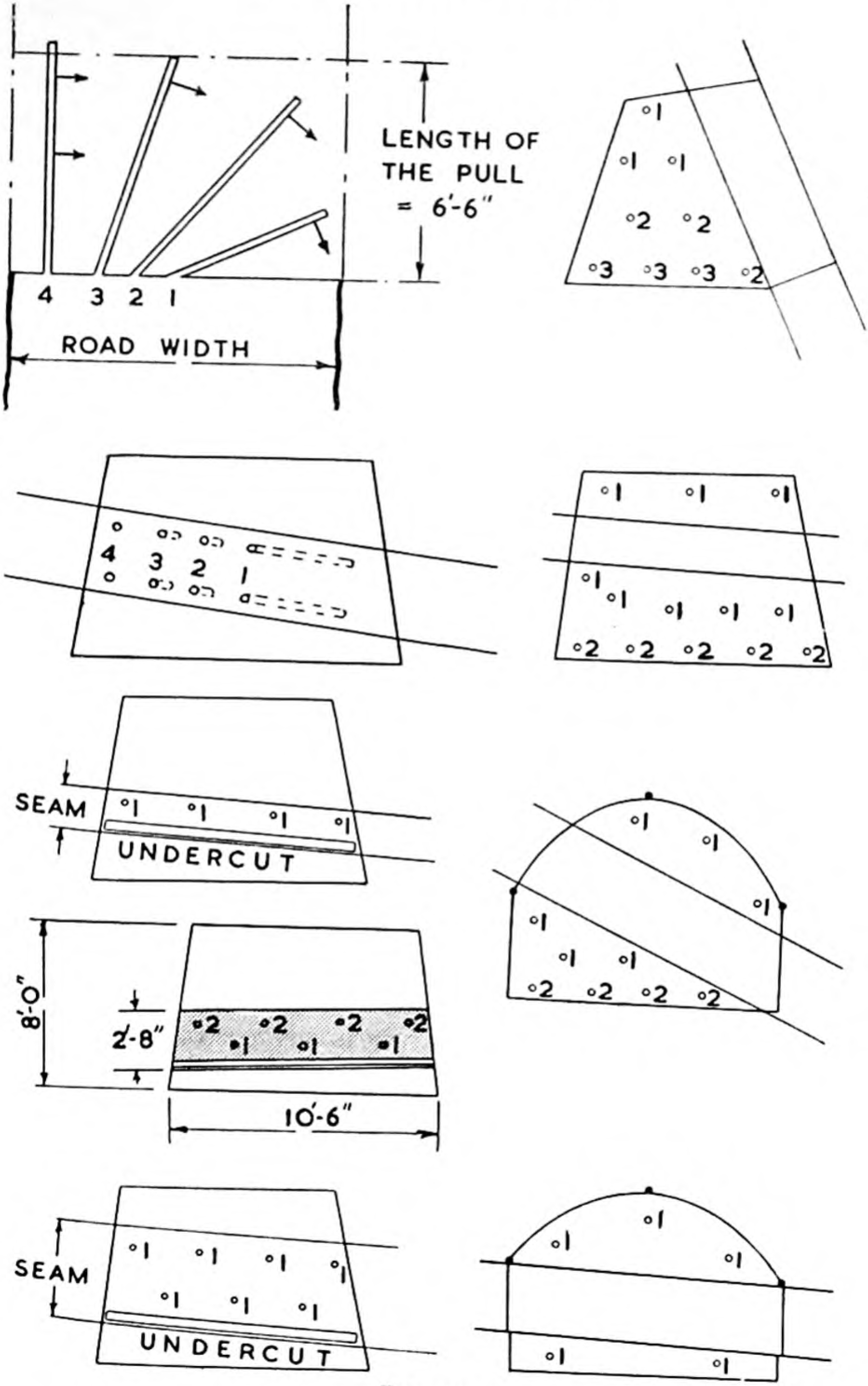

LENGTH OF
THE PULL
= 6'-6"
4 3 2 1
ROAD WIDTH
SEAM
UNDERCUT
8'-0"
2'-8"
10'-6"
SEAM
UNDERCUT

FIG. 205

from four to ten. Several examples are shown in Fig. 205. Depending upon the gas emission from the seam, instantaneous or delay action detonators may be used with the normal permitted explosive. The holes are drilled, using rotary machines, the length of the lift being somewhat more than the distance required between a pair of support sets (4 feet 6 inches to 6 feet). The greater length of lift is usual, since it is more economical to reduce the frequency of changes in the cycle of operations, such as drilling, charging, firing, etc. A shorter lift may be an advantage in obtaining an easier turn round between the operations and a better drifting performance, but the conditions will decide the best length to be attempted.

In the case of soft coal or rock, firing can be eliminated and the drivage done with pneumatic picks. Where the gate roads follow with the advancing face, the coal is worked across the gate on the face, the ripping in the gate roads being made easier by the sump left from coal extraction. The holes are drilled parallel with the road axis with rotary drills, using either carbide-tipped or hard metal bits, but with hard rock carbide bits are generally used. The number of holes depends upon the nature of the strata, and varies between two and fifteen. The length of holes can be based on 2 feet 3 inches to 3 feet 3 inches per cubic yard of rock to be fired. Fig 205 illustrates drilling pattern examples and the order of firing.

Section 3. Loading and Haulage

The coal and waste must be loaded out separately, the loading normally being done by hand.

Where the gate roads are driven with the face, the coal from these gate roads is loaded with the coal from the face. In advancing long-wall, where the gate roads are driven in advance of the face line, some means must be provided to transport the coal and stone the 10 or 20 or more yards back to the gate haulage. The coal from the advance gate can be loaded out only during the coal-filling shift on the face. If the coal is transported by car haulage, the coal from the advance gate is filled into tubs and taken out with the coal from the face as required. Where belt conveying is installed, the gate belt tension end is usually in advance of the face and extends into the advance gate road so that the coal can be filled out. The tension end and return box are covered with an appropriate guard during firing. Where the gate is ripped at the floor, a short conveyor from 5 to

10 yards long can be used with good effect. The conveyor can be either a belt or scraper chain, the belt conveyor being easier to dismantle and reassemble. Short shaker conveyors also can be used, and an example is shown in Fig. 206.

Coal haulage from gate roads which are driven independently of the face in retreating panel development can be carried out, using tubs or conveyors. In the latter case, the system is justified only if the road is to be used as a coal-haulage road in which the belt conveyor will continue to operate after the road is driven. Loading and haulage of the ripping material differs only from the methods applied for coal haulage, according to whether the waste from the ripping is

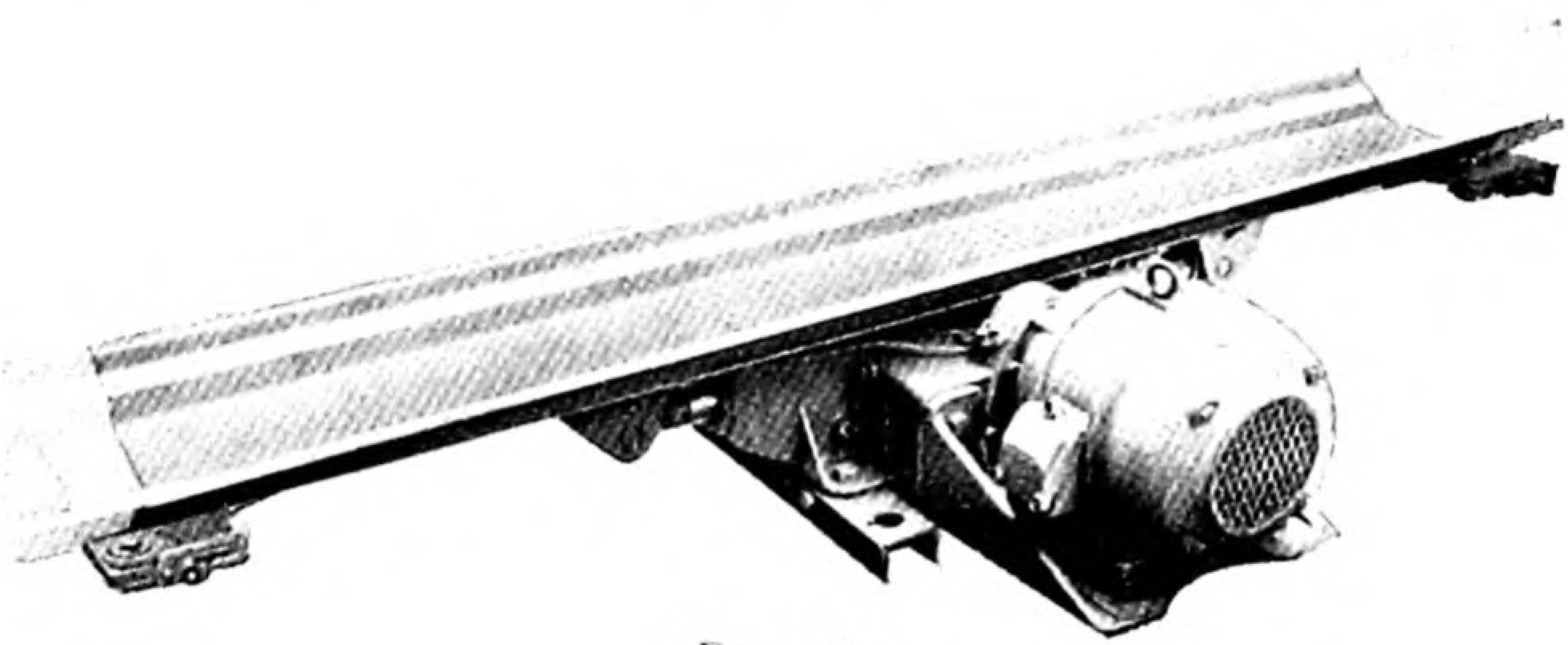


FIG. 206

to be stowed on the spot or transported outbye. Where the gate roads are up to the working face, the ripping is generally stowed either on the face or in a separate special pack on the rib-side of the gate. Where the seam is thin, and the amount of ripping material is too great to be stowed on the face alone, a gate rib-side pack can also be used. Where the seam inclination is more than 5 to 8 degrees, the waste is filled into a rib-side pack on the dip-side of the gate where it can be stowed more easily. In the case of steeper seams, the length of the face on the dip-side of the lower gate may be such that an auxiliary conveyor is required, a short scraper chain for the coal and a shaker or chute for the waste. The length or width of the pack varies between 4 and 15 yards. The ventilation of these fast-side ends, on the dip-side, is carried out by leaving small cross-gates in the pack or leaving a rib-side ventilation road, details of which are shown in Fig. 208.

The packs have the advantage of providing stowage space for the waste and assisting strata control by acting as an abutment pack for taking the main roof load from the gate road. The packs have the disadvantage, however, in that the adjacent face cannot use the gate for a main- or tailgate unless the pack is removed as the adjacent face advances, which is impracticable. For any adjacent longwall working a new gate road has to be driven alongside the old pack.

Where the gates are being developed from 10 to 20 yards or more in advance of the face, generally the waste ripping is loaded and taken outbye, or a small proportion may be stowed in a narrow roadside pack in the gate. The haulage used is the extension of the maingate haulage, either belt conveyor or car haulage, and in the latter case the waste can be loaded and hauled out during any shift. With belt haulage, the waste is conveyed outbye during a time when the coal face is not in operation.

With gate roads driven independently of the face, where retreating panels are being developed, the waste haulage is the same as for the coal.

Mechanisation of gate-road drivage has been recently attempted. This is more difficult than with stone drifts, since two kinds of material have to be loaded out and hauled during separate operations. The normal size of gate roads does not allow the adoption of the loading machines used at present for stone drifting. In some instances, ordinary and reciprocating shovel loaders are being used (Chapter 3, Part I, Section 2). In roadways where the cross-sectional area is larger, and the inclination suitable, the Joy Loader or a rocker shovel loader may be installed. The duckbill loader also has been used with some success.

The sectional area of the gate-road depends mainly on the means provided for coal and material transport and on the final length of the gate. The longer a road has to be, the longer it must be able to take the effect of strata settlement. The gradual decrease in the area of the gate due to the strata pressure and the resulting crush has led to driving roads of a greater area than required to compensate for this loss in area. Therefore, the roadway size required for adequate ventilation is attained in most cases and is often greater than necessary. In the case of a roadway for belt conveying, from 80 to 100 square feet is usual, while for car haulage, and where a double track is advisable for efficient running, the roadway must be large enough to take mine cars of 1 to 2 tons capacity. With larger mine

cars, the roadway section, in consequence, has to be increased.

With gate roads in seams of moderate thickness, the required height can be obtained by ripping the roof or floor, or both. The strength of the roof and floor strata and their ability to withstand pressure as well as the ultimate use of the road will decide the best procedure to adopt.

Where the roof and floor strata are different in character, the softer rock will be ripped. With similar roof and floor strata in level measures it will be preferable to rip the roof, since loading out will be easier, especially into roadside packs on the face. With seam inclinations of more than from 3 to 5 degrees, it is recommended, in any case, to rip the floor, at least to such an extent as will provide a level working floor advisable for the haulage.

The purpose the road has to serve will affect the thickness of the ripping, since in coal-haulage roads sufficient difference in height between the seam level and the floor in the gate is necessary to allow the face conveyor to load the coal into the gate haulage without difficulty. In the case of conveyor haulage on the gate road, the difference in level may be between 15 and 36 inches and for car haulage 3 feet to 4 feet 6 inches, although the use of a gate-end loader can obviate the need for direct loading from the face into the mine car.

Section 4. Organisation and Performance in the Driving of Gate Roads

Normally three men are employed per shift at the face of these headings and, in view of the available working space and the distribution of work, this is found to be the best number to give the optimum average rate of advance per manshift, both at the face and overall. With a smaller number of men the overall advance decreases, and while it is true that the greater number of men give a higher total advance, the rate per manshift drops progressively. This is not the case where the gates are being ripped up to the working face, or in gate roads ahead of the face where the ripping is being stowed in a gateside pack. In these cases, one or two additional men can be employed during the packing shift as pack-men.

When organising the cycle of operations in the normal gate roads brushed up with the face, care has to be taken not to interfere with the coal working cycle on the face. The loading out of the ripping material must be done when coal production is not taking place.

In the following examples the labour employed and cycles of operations required are discussed for gate roads under different conditions:

Examples of the organisation of gate-road drivage

Case 1

Gate-road Details

Type . . .	Belt conveyor road in line with face.
Excavation . .	105 square feet.
Lift . . .	4 feet 6 inches.
Seam Thickness .	4 feet 6 inches.

	Morning Shift	*Afternoon Shift*	*Night Shift*
Coal face.	Filling.	Shifting up conveyor.	Packing.
Gate road.	Supports. Setting one to two supports.	Ripping roof and packing waste on both sides of roadway.	Ripping floor and setting two dirt chocks. The ripping waste is packed into the chocks.
Labour employed	2	4	4

Case 2

Gate-road Details

Type . . .	Belt conveyor road in advance of face.
Excavation . .	90 square feet.
Lift . . .	4 feet 6 inches.
Seam Thickness .	6 feet.

	Morning Shift	*Afternoon Shift*	*Night Shift*
Coal face.	Packing.	Filling.	Shifting up Conveyor.
Gate road.	Ripping roof and floor, inserting roadside packs. Setting door frame supports.	Coal filling out of advance heading.	None.
Labour employed.	3	2	

Case 3

Gate-road Details

Type . . .	Belt conveyor road in advance of face.
Excavation . .	80 square feet.
Lift . . .	6 feet.
Seam Thickness .	2 feet 9 inches.

	Morning Shift	*Afternoon Shift*	*Night Shift*
Coal face.	1st filling shift.	2nd filling shift.	Shifting up conveyor. Packing shift (caving).
Gate road.	Setting two supports (polygonal steel supports).	Filling out coal from heading in advance—3 men. Drilling holes in roof ripping and firing round at end of shift —2 men.	Filling waste from heading and ripping floor—4 men. Erecting two dirt chocks —2 men.
Labour employed	3	5	6

Section 5. Comparison of Gate Roads driven in Advance of the Face and driven with the Face

A generally accepted argument deciding clearly and definitely in favour of either method of development cannot be given. Both methods have their advantages and disadvantages, and each individual case must be based on the prevailing operational and geological conditions to decide whether the favourable or unfavourable considerations are decisive.

The main problem with gate roads in advance of the working face is one of support, while with gate roads brushed up to the face it is a question of gate-road haulage. It is usual to have the main gate in flat measures, and the lower gate road in inclined measures, acting as the conveyor or loading gates driven ahead of the face. In the case of tailgates or upper gate roads, particularly where it is not necessary to bring in waste, these follow with the face. The greater the thickness of the seam, the more usual it is to keep the main gate road in line with the face.

The advantages and disadvantages of forward gate-road development are reversed in the case of gates in line with the face.

Advantages of gate roads in advance:

1. A limited exploration in advance headings in faulted areas is an advantage, since the later extraction can be varied to suit the conditions discovered in advance of the face.

2. The transfer from the face conveyor to the centregate or main-gate conveyor is easier, especially in thin seams. There is less danger of damage to the main conveyor with consequent breakdown of the gate-road haulage system due to dropping the ripping on to the conveyor structure.

3. Easier organisation of the gate-road work is possible, since it is more independent of the general face work than in gate roads brushed up to the line of the face.

Disadvantages of gate roads in advance:

1. There is a lower rate of progress in gate roads in advance of the face, since in gate roads brushed up to the face the coal is extracted with the face. The thicker the seam, the greater is the difference in performance in favour of gate roads brushed up with the face.

2. There are higher maintenance costs, since the supports are set in a forward abutment area and they suffer the effects of this initial loading and the back abutment loading as the face advances.

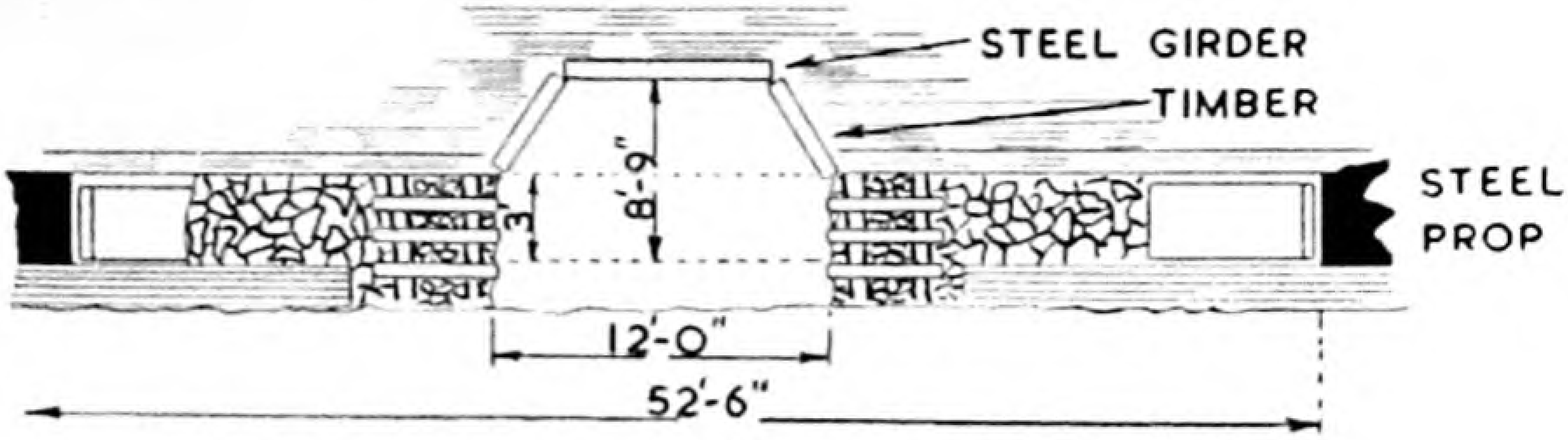
3. Temporary supports are frequently required and permanent packs are needed on the face side of the gate with the usual provision of wooden chocks or filled chocks.

4. Drivage cost is higher due to auxiliary ventilation where this is required.

5. The consumption of explosives is higher since the coal in the heading in advance is under the pressure of the front abutment and is harder to break down. The adjacent rock is also harder to break down than in gate roads following the face, since a certain loosening of the strata has already taken place due to the coal extraction.

Section 6. Driving Gate Roads with a Short Face Development

The gate roads may be developed in such a manner that a short coal face is set out from 20 to 30 yards long. The gate road which advances with the face is ripped, and the waste stowed in a pack on



SECTION ON A-B

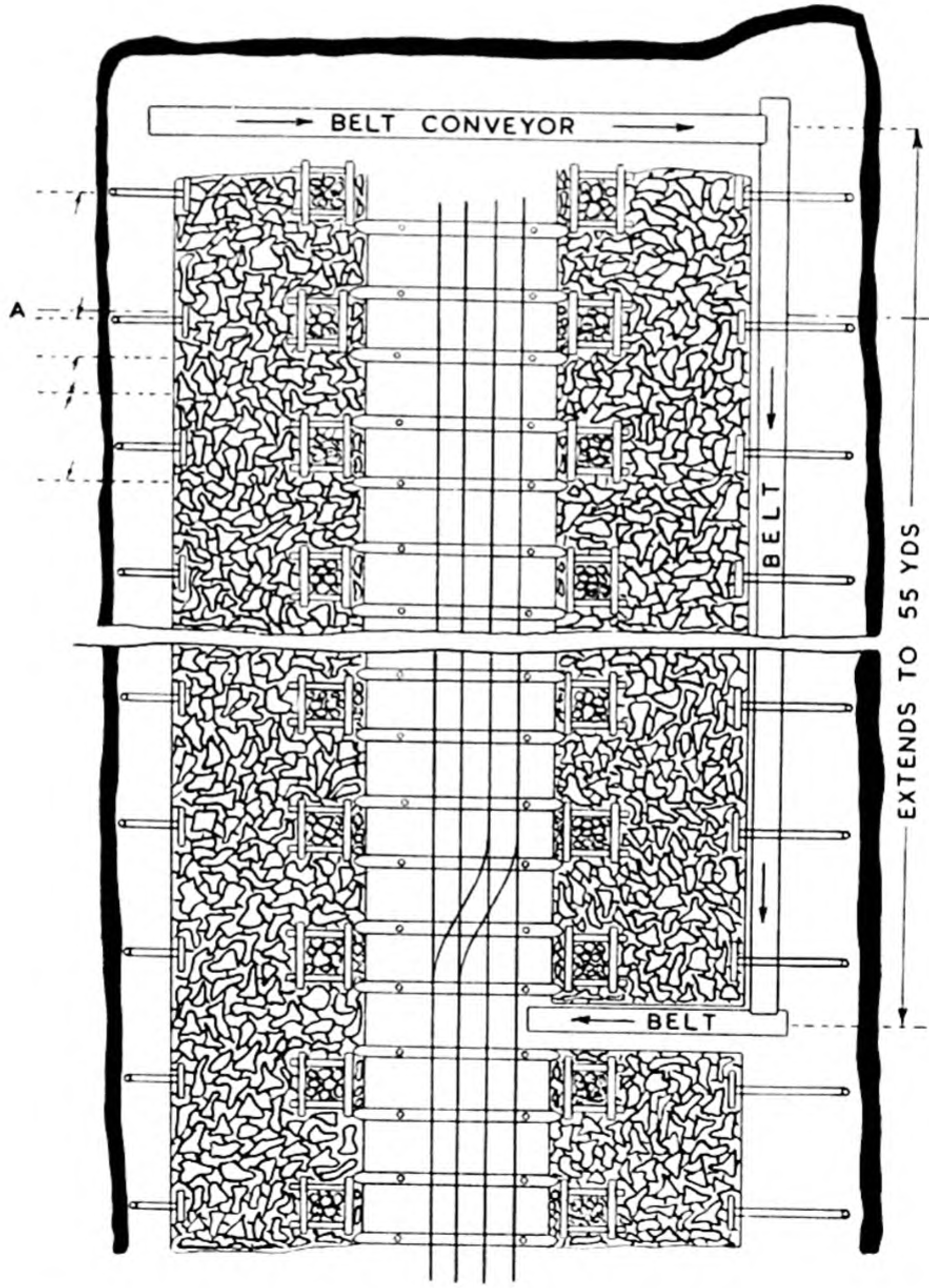


Fig. 207

the short face. This system has the advantage of a high rate of advance, probably 3 or 4 yards per day. This is achieved by organising the operations of coal-getting, brushing, coal haulage and the setting of supports so that they can be done at different points and to a large extent simultaneously. The pack assists in providing an area where the transverse abutment loading can be transferred away from the actual gate road.

There are several variations in the general layout and organisation of the system. One method is illustrated in Fig. 207, which shows a layout having a central main gate road, on each side of which packs are built. The face is 17½ yards wide, the main gate road being 4 yards wide. The barrier gates of the face are driven at seam height, the right side road being equipped with a belt conveyor for coal transport from the face. This belt conveyor can be extended up to 50 yards and delivers on to a short belt conveyor installed in a stenton, or break-through, in the pack, from which the coal is fed into cars in the gate-road. The layout in Fig. 208 is an example of a scheme in which the gate-road is driven with a pack on one side only, the pack being built from the ripping material from the gate roads. The small rib-side road is kept at seam height for ventilation purposes. The main feature of the layout is that the short face advances with the main face.

A further example from British practice, which is commonly termed a 'self-stowing heading', is shown in Fig. 209. This layout varies from the central gate-road system only in minor details. A gate-end loader is used, filling directly into mine cars, and the face of the main gate road is free from coal transport, so that it can be drilled and fired during the face advance. The gate road serves also as adequate car storage, from which the cars are run under the gate-end loader.

The organisation and distribution of the personnel in the case of the central gate-road layout (Fig. 207) may be carried out as in the following summary:

Type	Haulage gate driven in conjunction with short face.
Excavation . .	78 square feet.
Lift	6 feet.
Thickness of Seam .	3 feet.

	Morning Shift	*Afternoon Shift*	*Night Shift*
Coal face.	Filling.	Shifting up conveyor.	Packing.
Gate road and including short face.	Filling off short face—5 men. Drilling roof and floor rippings—firing rounds—2 men.	Packing waste from heading on both sides of road — 4 men. Shifting up conveyor on short face—2 men.	Setting two dirt chocks and two supports (door frame supports on top of gate packs).
Labour employed.	7	6	3

PART II

METHODS OF SUPPORT IN GATE ROADS

Section 1. *General Introduction*

In comparison with drifts driven in solid rock strata, the gate road driven in an area where coal has been, or is being, extracted is subjected to roof and floor convergence and must have a method of support to allow for a continuing amount of convergence and capable of controlled yield.

Where the support is not capable of giving sufficient yield, the supports are damaged and the surrounding rock strata fractures and tends to cave into the roadway. It is well known that the roof bends over and into the roadway from the sides towards the centre. The lateral pressure from the sides affects the roof and adds an additional load to the supports.

The resistance or yield of the gateway pack-wall material are also important factors. The gate-road walls can either be packs, timber chocks or solid rib-sides. Should their capacity to resist the strata pressure be greater than the waste fill in the adjacent goaf, the main roof load will be taken on the more resistant abutments and a concentration of roof stress will occur around the excavation forming the roadway. Due to this stress concentration, the floor as well as the roof strata are affected. The additional stress is transmitted by the resistant road sides to the floor of the seam. Where the floor is soft, as is often the case, movement is started, the floor of the roadway is lifted up and 'heaving' occurs. The objective is to avoid the additional pressure zones in the immediate vicinity of

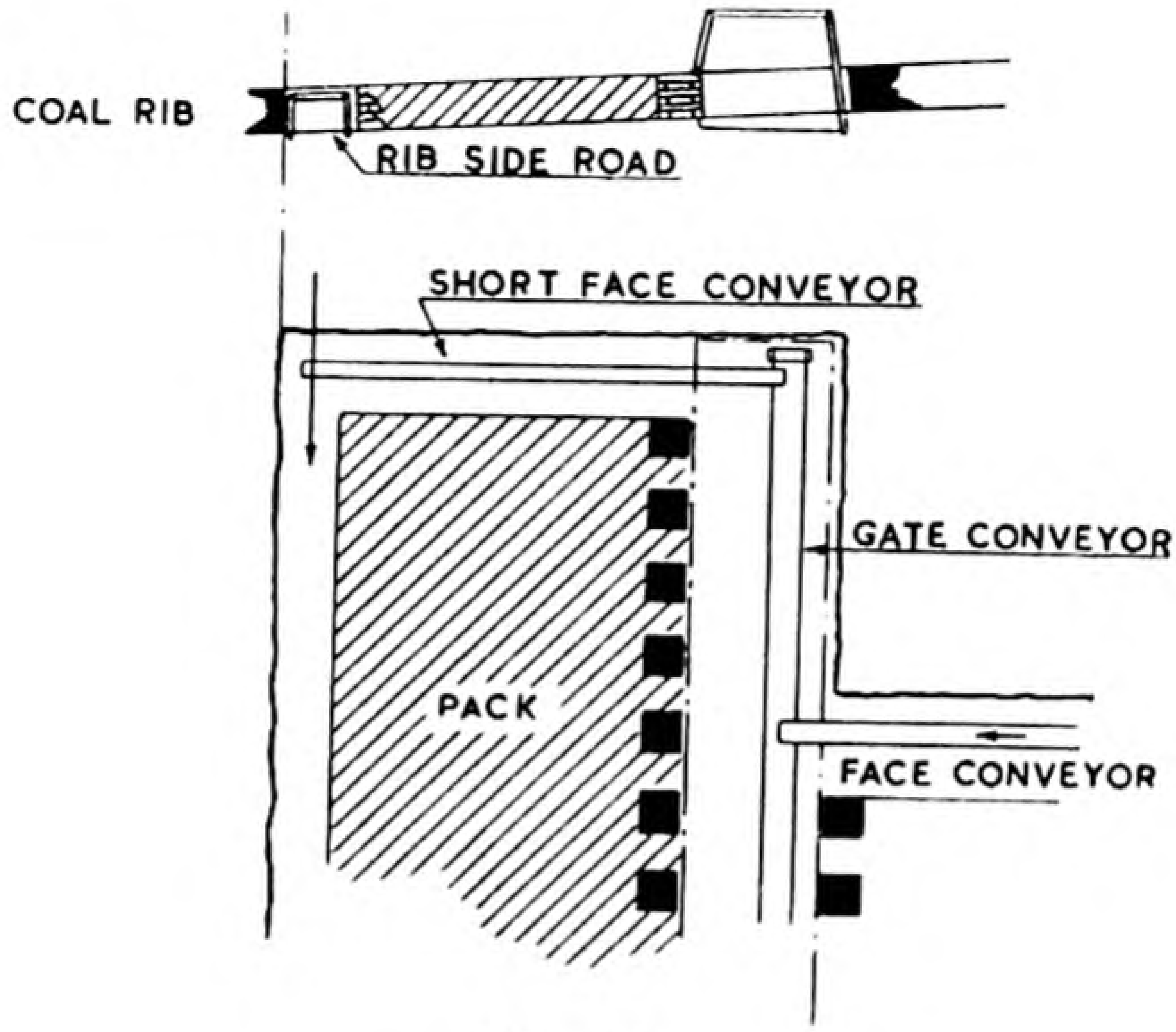
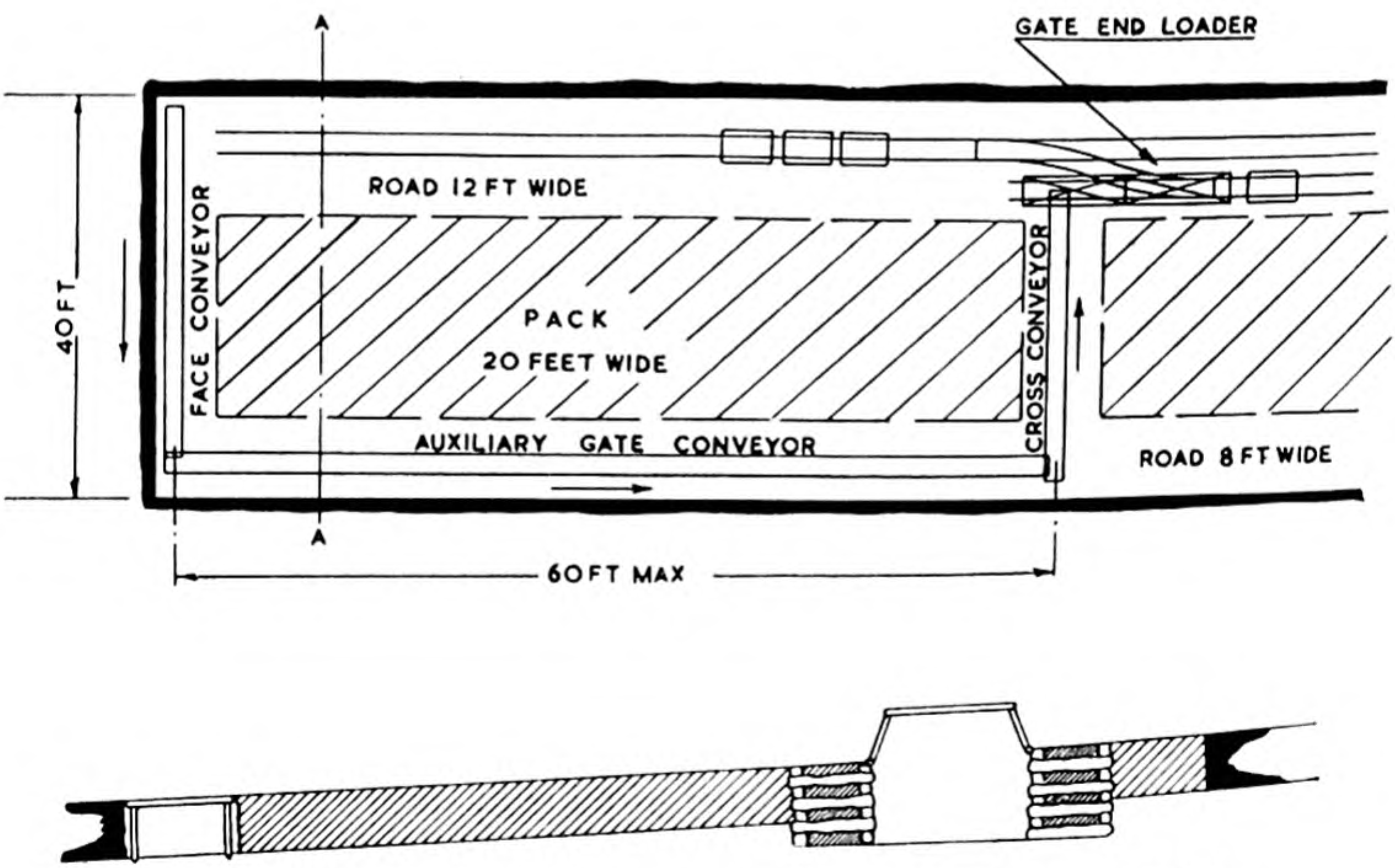


Fig. 208



SECTION ACROSS FACE AT A-A

Fig. 209

the roadway by deviating the load from the road into the roadside packs or, better still, into the goaf, or, at least, taking care to distribute the load uniformly over a larger bearing area.

Where the gate road is driven with coal sides, there is sufficient possibility in the case of a soft coal and a hard rock strata to transfer the pressure, and the road side coal then acts as a yielding border. A pack is still considered necessary in order to reduce any pressure effects on the roadway roof.

Where the gate road is driven with a goaf on one or both sides, the packs or chocks adjacent to the road should have at the most the same resistance to crushing as the waste filling on the face. Thus, for total caving, the lateral packs should be wide and allowed to yield. Generally, this means that the roadway sides are considered as essential parts of the roadway support, and their yield should be related to the packs used and the stability of the roof and floor. Another method used to transfer the roof pressure from the gate-road is by parallel pressure relief gates left from 6 to 10 yards apart on each side of the gate road. The cheapest and simplest form is by means of scourings left in the waste pack on the face. These small roads absorb the roof pressure (the roof breaking down into them) and do not serve any other purpose.

The yield in the gate-road supports must be arranged in such a way that initially they yield quickly and finally provide a greater resistance. The initial yield should not be provided by any weakness on the part of the supports, but by some method of control. Generally, lateral movement does not exert a great stress on the supports and can be absorbed effectively without difficulty.

Finally, a gate road in the goaf can be maintained easily after a certain period of time has elapsed, corresponding to an advance of 100 yards of the winning face. By this time the main roof movement has ceased and the back abutment has moved forward past the area in question. The length from the face to a point 100 yards outbye is the section requiring the highest maintenance cost, while farther outbye the maintenance cost is generally a great deal less.

Section 2. Timber Supports in Gate Roads

Timber supports are relatively cheap, but have a low resistance to roof pressure. The increased dimensions of roadways over the past few decades, together with the greater demands for efficient

roadway haulage, have proved that timber is no longer in a position to meet these requirements.

Timber is used in certain appropriate cases where, for instance, it is not necessary to renew the supports more than once and the maintenance costs are low.

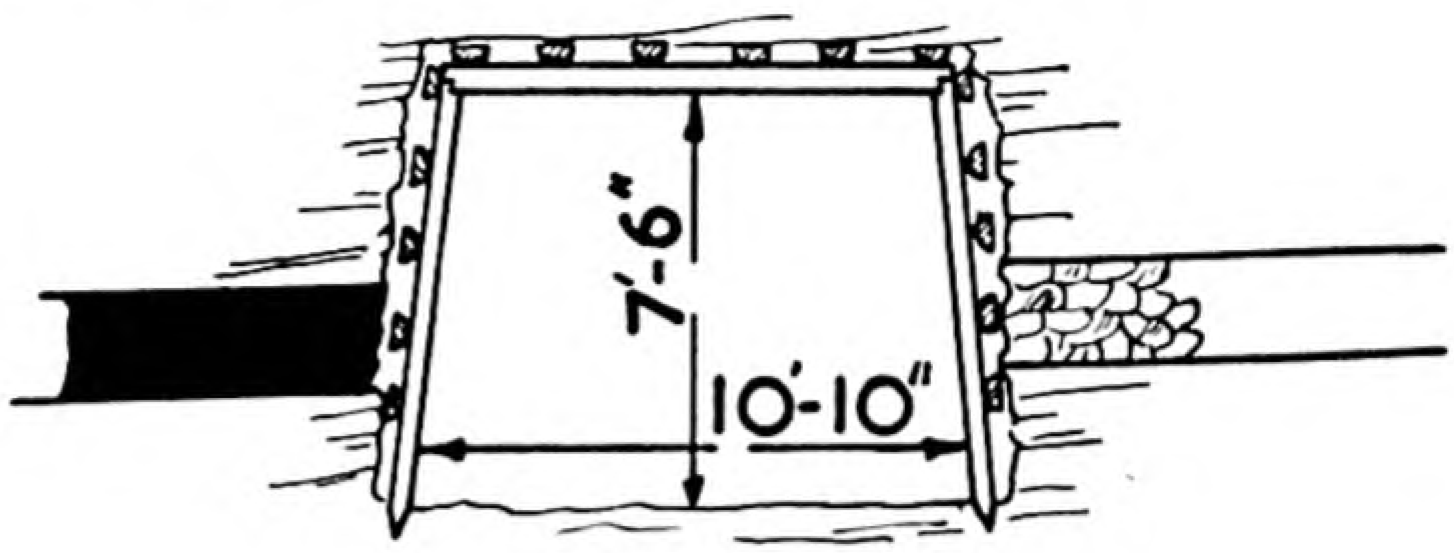


FIG. 210

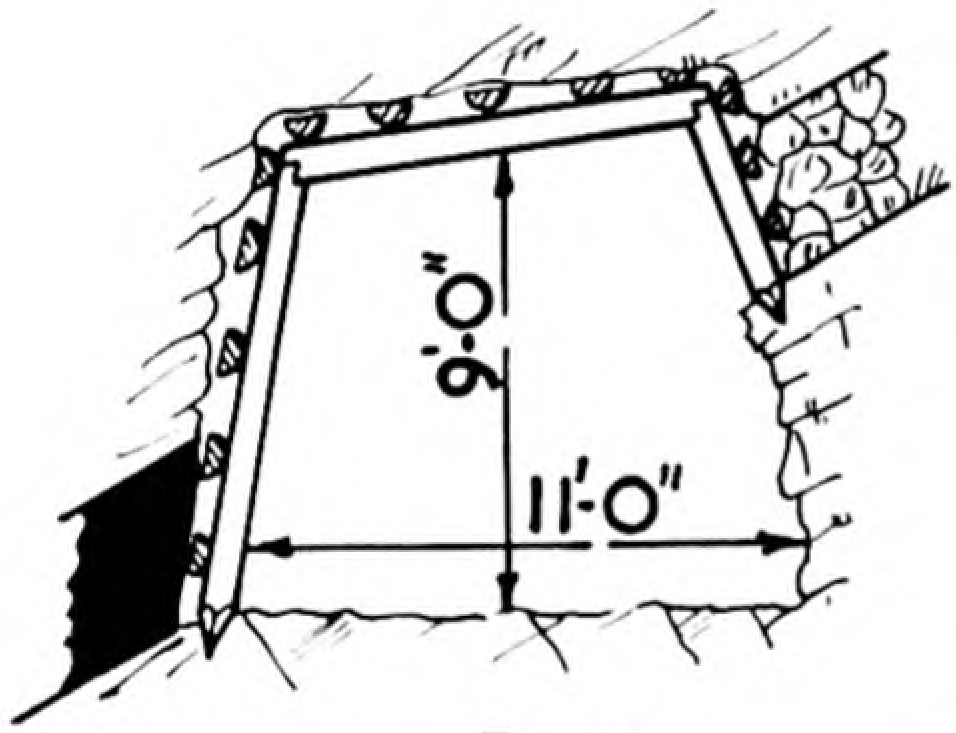


FIG. 211

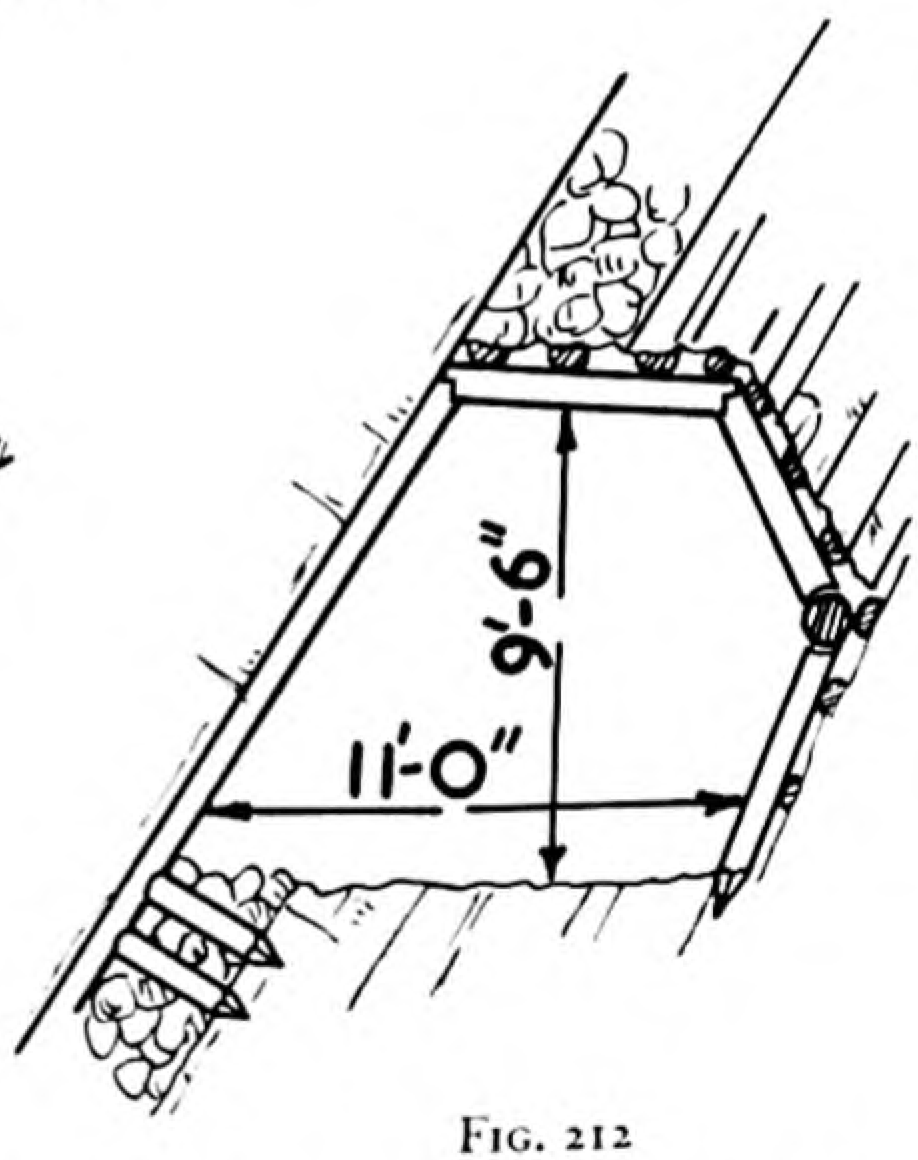


FIG. 212

The commonest form is the doorstead support, which may be used as a complete or as a curtailed doorstead frame. Figs. 210, 211 and 212 show examples of doorstead supports in flat, semi-steep and steep seam conditions. In order to increase the yielding properties of the support, the doorstead legs are tapered. Timber cushion pieces are required on the cap timber in the direction of the prop axis, as well as at the sides of the roof timber. Good lagging and tight packing of the space between the lagging and the roof are essential, and the walls also should be lagged. As in stone drifts, the supporting frames should be well strutted between the individual

frames or sets. The interval between the sets depends upon the type of strata and varies from 2 to 4 feet.

In order to increase the yield, curtailed doorstead frames are used and, as shown in Figs. 213 and 214, are set on chocks. Care must be taken during the setting of the frames to ensure that the props of the frame are resting on the corners of the chocks. The chocks have the advantage of providing a wide area of support to the roof, while being able to yield. The chocks can be dirt-filled and the base is

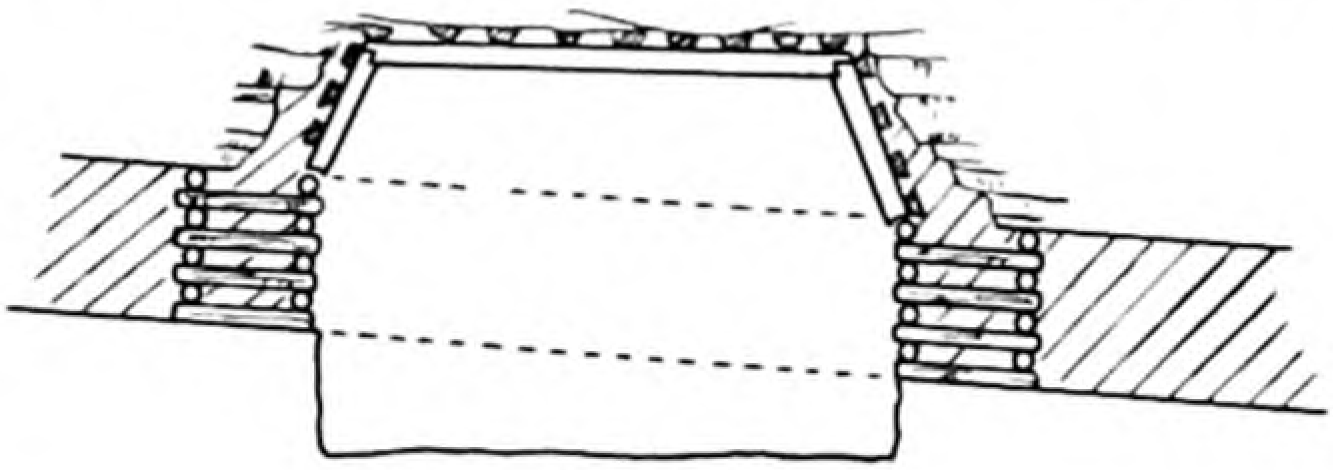


FIG. 213

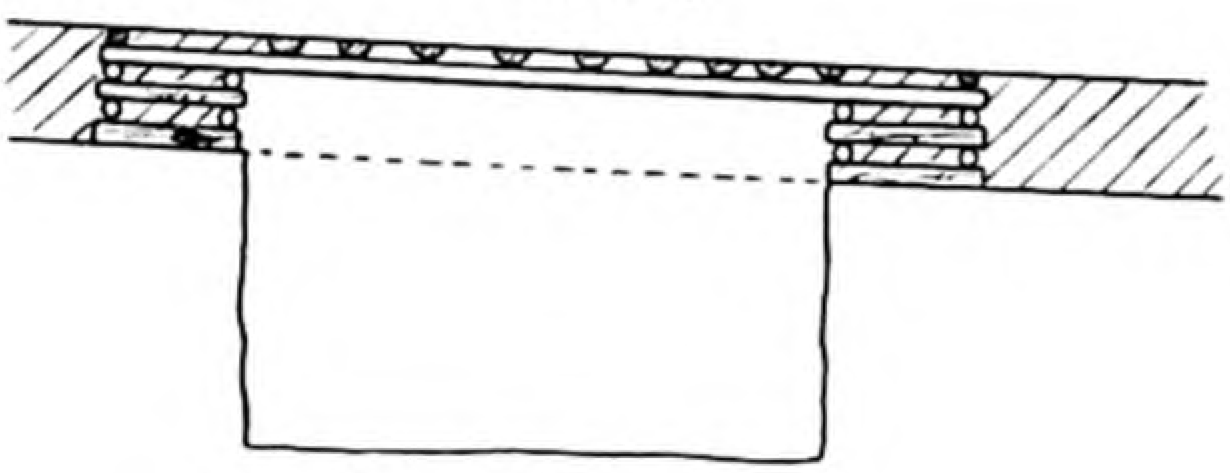


FIG. 214

usually 4 × 4 feet. The chocks are set either close together or at intervals of a chock width apart. The high consumption of timber is a disadvantage and, if possible, old face timber is used for the chocks. It is also possible to use stone walls with wooden cushion pieces. The cushion pieces are important, as they provide the desired yield, while without them the pack wall would be too rigid and would allow the floor to heave. The pack-wall material should be strong shale or sandstone; soft shale is unsuitable.

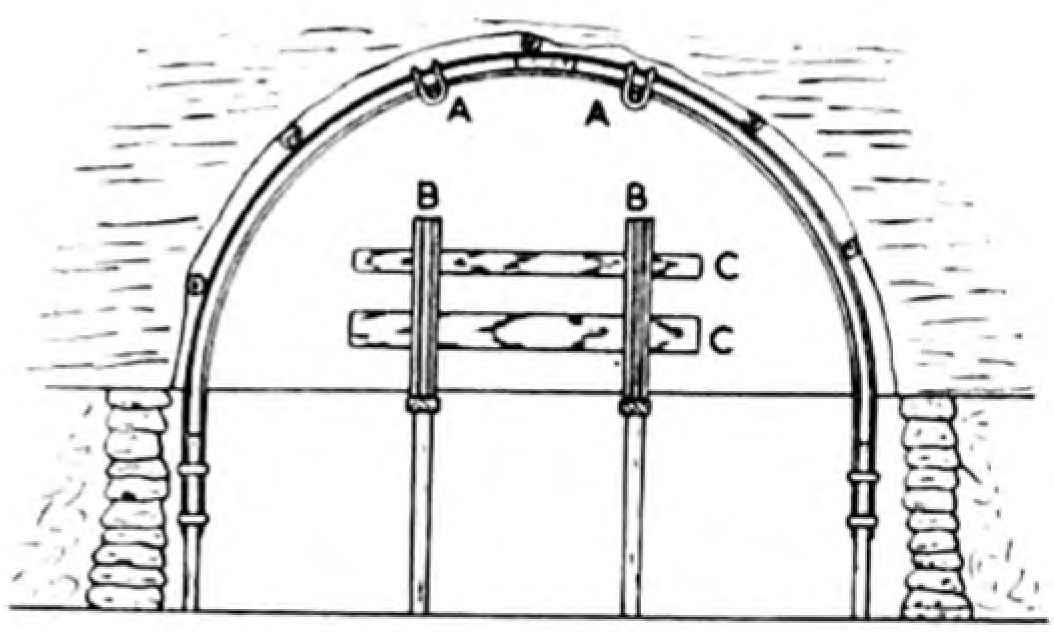
Where the roof of the seam is acting as a working roof, as may occur in thick seams, or when taking a floor ripping under good roof conditions, it is usual to leave the doorstead legs out and to put the cap-pieces immediately on the chocks, *vide* Fig. 214.

The weakest part of the frame is the cap-piece, and consequently it suffers most from the roof pressure. To increase its supporting capacity, a steel strap may be used instead of the wooden cap-piece. The strap can be used as part of the complete or curtailed doorstead

frame, and it is possible to use it straight on to the chocks. The strap can be withdrawn only with difficulty, or not at all when the gate road is abandoned.

Section 3. Steel Supports in Gate Roads

The steel sections used for gate-road support are the same shape, type and quality as used for stone drifts, *vide* Chapter 3, Part II, Section 2. It should be emphasised, however, that the steel supports in gate roads must be of the yielding type, the yield being obtained



A. HANGERS OVER STEEL ARCHES.
B. CORRUGATED STEEL STRAPS.
C. LAGGING.

FIG. 215

by a combination of steel and timber or included in the steel-girder framework design.

The steel arch pattern is well known, consisting of two segments connected by fish-plates, *vide* Chapter 3, Part II, Section 2(*e*). The legs are mounted on wooden or steel stilts in order to provide the necessary yield as shown in Fig. 215, the stilts being set inside the web so as to telescope within the arch. This figure is a cross-section of the gate road viewed in the direction of the face, B and C showing how the ripping lip is supported.

The polygonal type of support has proved successful in practice and has been used in stone drifts, *vide* Chapter 3, Part II, Section 2(*g*). In gate roads the steel supports are usually of a lighter section and the legs are set on wooden chocks, giving a higher supporting capacity and better yield. Figs. 216 and 217 show two types of construction, while Fig. 218 shows a polygonal arch set on timber props. In semi-steep and steep conditions, this type of support must be installed carefully so that the joints are against the rock, on a solid

FIG. 216

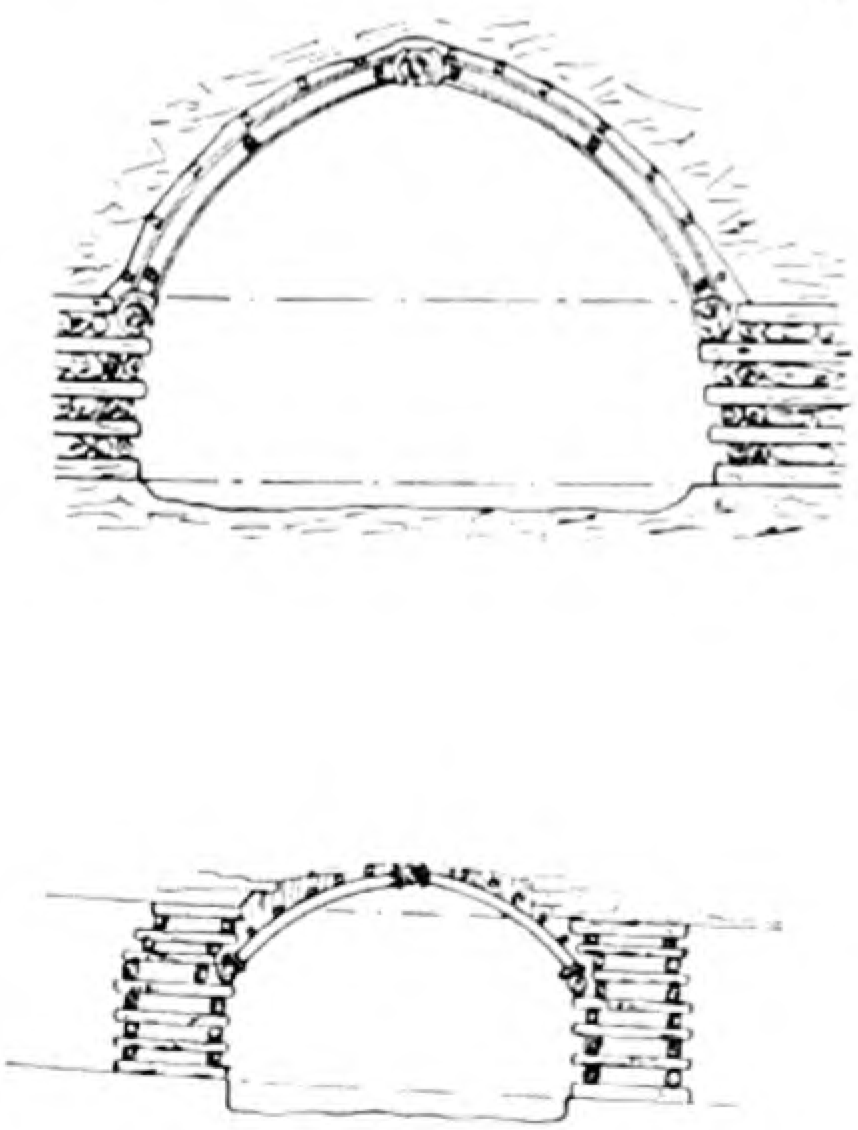


FIG. 217

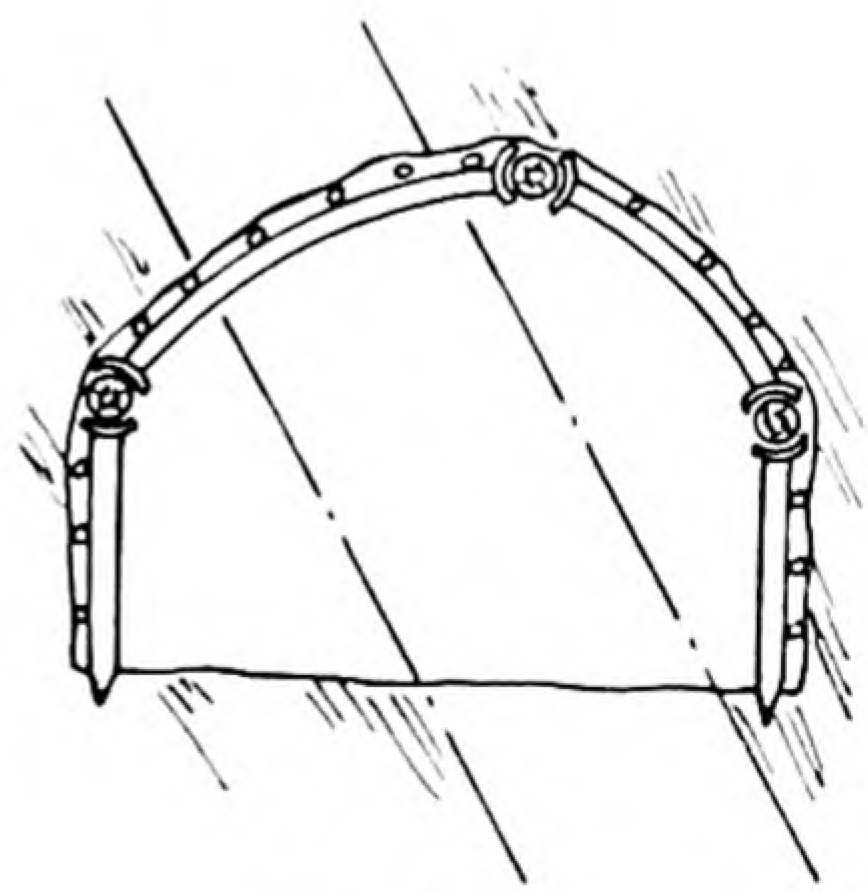


FIG. 218

base, and not against the coal side. This may necessitate using segments unequal in length, as shown in Fig. 218. Since the polygonal support is not fish-plated and bolted, but set with loose

sections, the support must be set exactly vertical and the frames tied firmly to each other. A steel support in which the yield is provided in the design is the Toussaint-Heintzmann support, which has been described in Chapter 3, Part II, Section 2. Two types of this form of support, which are used more frequently in steep than in flat measures, are shown in Fig. 219. Another type of yielding support,

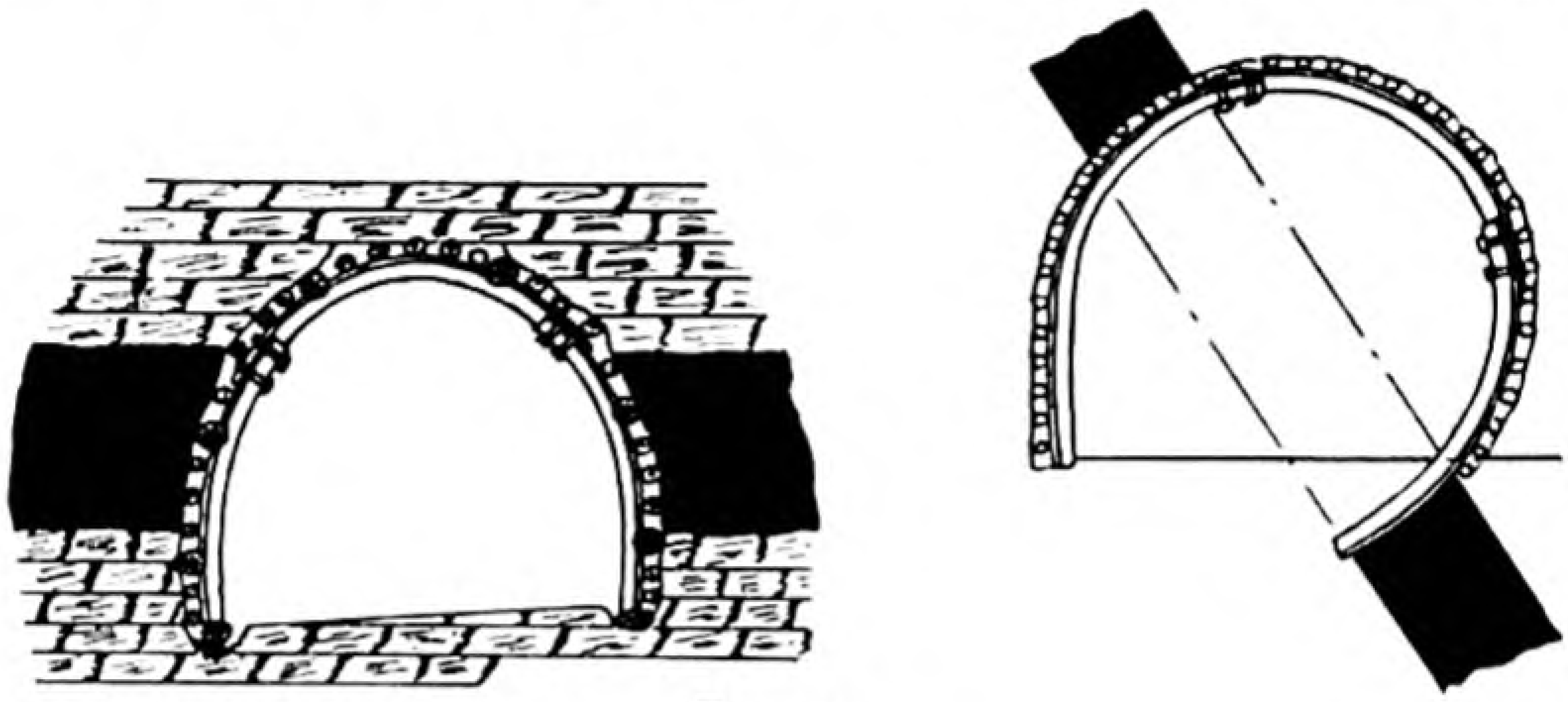


FIG. 219

specially fit for use in steep or semi-steep seams, is characterised by a single yielding segment of the Toussaint-Heintzmann type between roof and floor, the other segment being rigid. In some cases, roof bolting has proved useful.

PART III

DRIVING RISES OR INCLINED ROADS IN COAL

Section 1. *General Introduction*

An inclined road or 'rise' connects two or more gate-roads and is normally driven on the full dip of the seam. It is therefore at right angles to the gate roads, which are driven to the strike. In longwall development, the rise is driven to win out the face lying between the gate roads, and on completion of the rise, only requires the installation of the face conveyor in order to start face production. This procedure applies to either advancing or retreating faces.

As soon as the face commences, the rises disappear as roads and become part of the goaf behind the face. In some cases they may be retained for a time for ventilation and hoisting or as access roads to and from upper, lower or sub-levels. Where they are used for coal haulage, the height of the road must be arranged to take either belt or shaker conveyor transport, but sometimes rope haulage may be used. The amount of ripping to be taken depends upon the height

of the seam. If the road is only for face development, ripping is required only in very thin seams. Rises can be driven either to the rise or to the dip of the seam.

Section 2. Rises Driven against the Dip of the Seam

Three main operations have to be distinguished when driving rises, viz. the winning of the coal (and under certain circumstances ripping the road), the haulage of coal and material, and the setting of supports. These are described in separate sub-sections as follows:

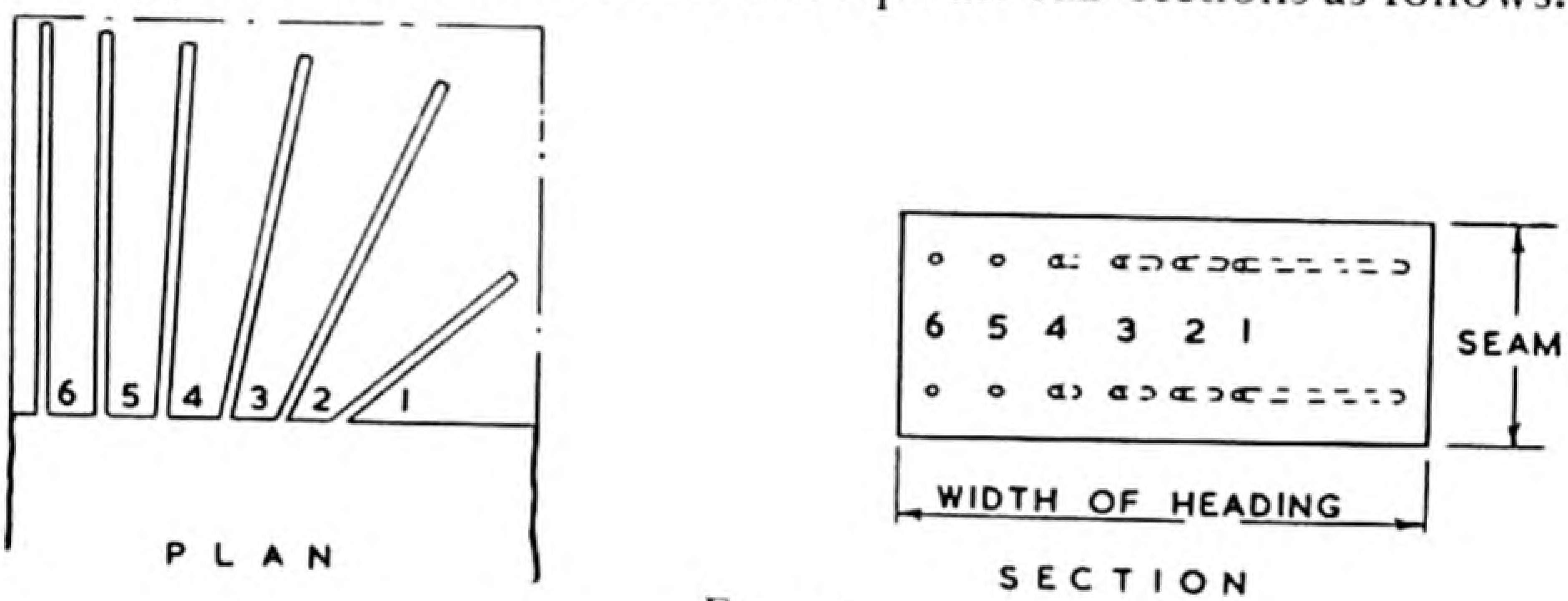


FIG. 220

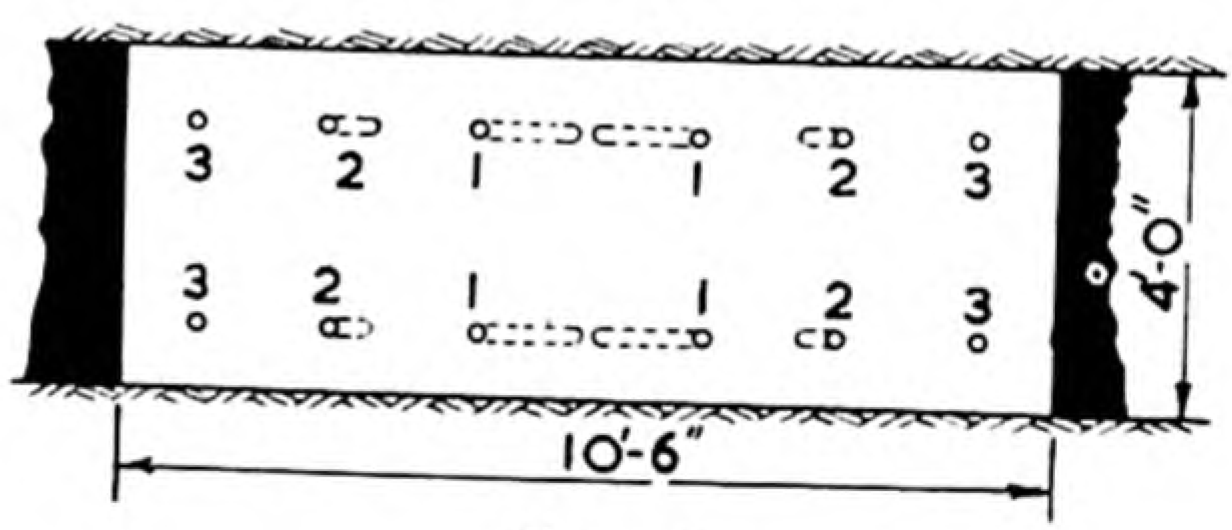


FIG. 221

(*a*) *Method of Working.* The method of working the coal depends upon the hardness of the coal. In British practice, generally the coal is drilled and fired with previous coal-cutting. Cutting need not be used if the coal is not hard and if there is no restriction on firing the solid coal. The drilling and firing are carried out as in gate roads, and examples of the drilling patterns and sequence of firing are shown in Figs. 220 and 221, from which it is seen that various drilling patterns are used. The cutters, or sumpers, can be arranged to give a wedge, cone or triangular cut. A fan cut can be introduced where this pattern will give the best result. The explosive used is the ordinary permitted explosive for use in gassy mines, and delay or instantaneous detonators may be adopted where their use is per-

mitted. The pull per round may be in the region of 2 yards. Short-wall coal-cutters are used where the coal is cut, and the machine setting and the arrangement of haulage ropes for undercutting are shown in Fig. 222. The drilling pattern for a face which has been under-cut is different from that when firing off the solid, since in the former case the holes are located to act towards the free face provided by the cut. The holes in this case are parallel to each other and in the direction of advance. In thin seams, a single line of holes may

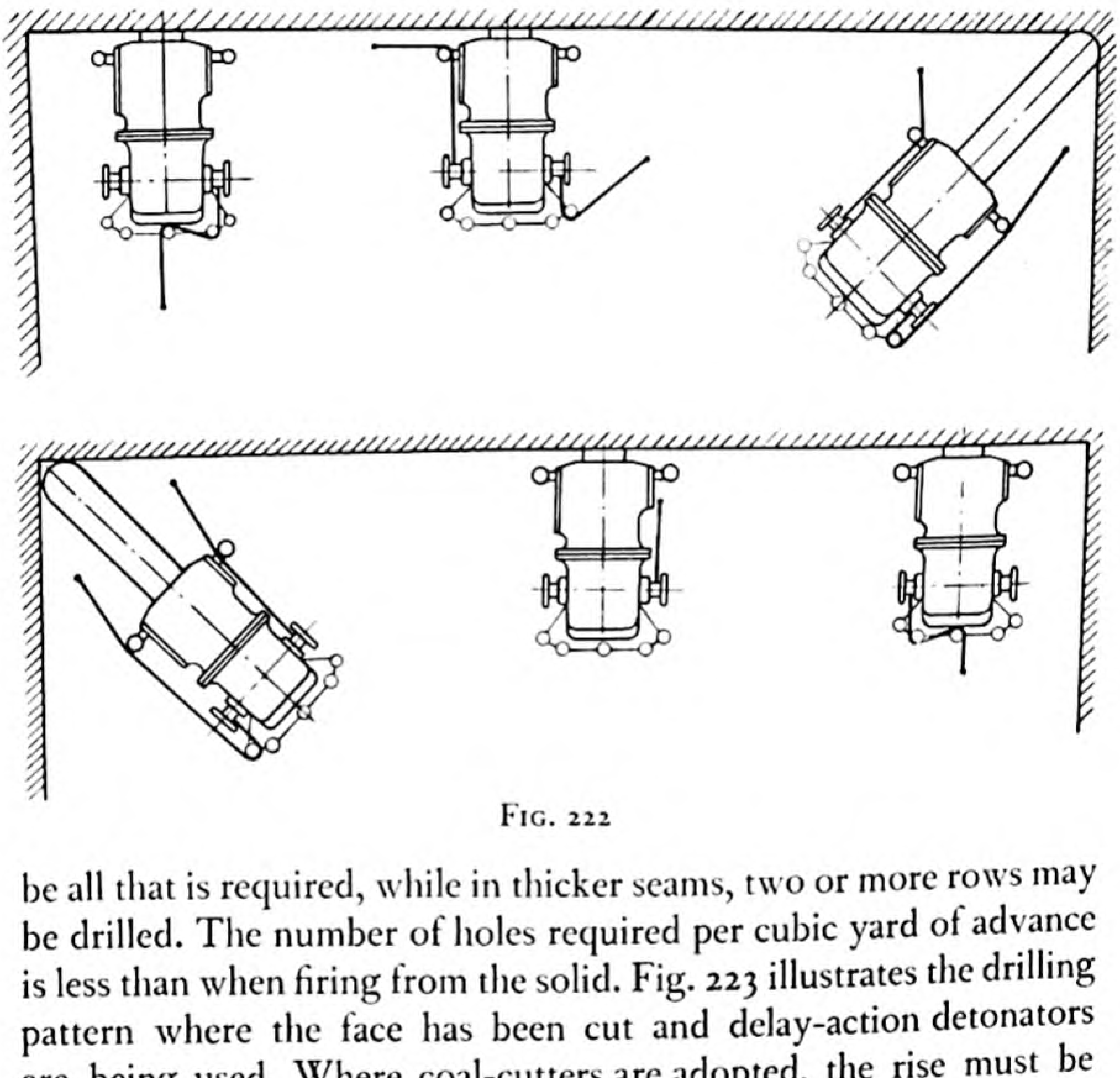


FIG. 222

be all that is required, while in thicker seams, two or more rows may be drilled. The number of holes required per cubic yard of advance is less than when firing from the solid. Fig. 223 illustrates the drilling pattern where the face has been cut and delay-action detonators are being used. Where coal-cutters are adopted, the rise must be sufficiently wide, about 4 yards, to accommodate the coal-cutter during loading.

Vertical shear cuts can replace the normal horizontal cut where this method has an advantage. Since the advantage of a shear cut over an under-cut is very slight in many cases, the extended use of such machines is not anticipated. Fig. 224 is an example of a rise face which has been shear-cut, showing the alteration in the drilling pattern and sequence of firing.

On the Continent, where the coal is usually soft, the rise can be driven, using the pneumatic pick. The coal may or may not be cut, depending upon whether this will assist the pick-operators in their task. One pick is usually sufficient. The use of two pick-men is not required unless the width of the rise is more than 3 yards, or the seam is thick, in which cases the coal is taken out in more than one lift, the first lift being taken out 2 yards in advance.

The rise must be ripped where it is necessary to make a larger roadway, higher than the normal seam height, as may be the case where the rise is to be used for haulage or ventilation. The nature of the roof and floor will decide the position of the ripping.

(*b*) *Haulage*. The conveyor to be installed in the rise can be any of the three usual types, viz. belt, bucket-scraper or shaker conveyor.

Where the inclination of the seam is less than 14 degrees the belt conveyor is recommended. The belt conveyor for rise development should be always of the reversible type, as compared with the normal belt conveyor usually installed in drifts. This type of belt conveyor, although expensive and more prone to mechanical trouble than either of the other types mentioned, has the advantage of being able to convey in two directions and thus serve for coal and materials transport. In order to avoid the daily lengthening of the belt conveyor, a short auxiliary belt or shaker conveyor can be used at the face to feed on to the main conveyor, with the main belt being lengthened in fixed distances of 50, 100 and 150 yards of drivage. A special arrangement for rapid lengthening of the belt conveyor has been designed and is in use on the Continent, *vide* Fig. 225. Such equipment becomes very important in this instance, where the daily advance may be 10 yards or more. The feeder conveyor previously mentioned can be lengthened easily each day in one shift by three men.

The shaker conveyor requires an additional means of materials haulage, but is lengthened as required by the daily advance. The shaker conveyor operates at its best on inclinations between 10 and 25 degrees; at steeper inclinations, up to 40 degrees, fixed chutes should be used. At inclinations greater than 40 degrees, the coal can be gravitated down a 'pass' separated from the rest of the rise by a wooden partition. When using fixed chutes (shaker pans), to prevent the formation of dust it is necessary to keep the chute

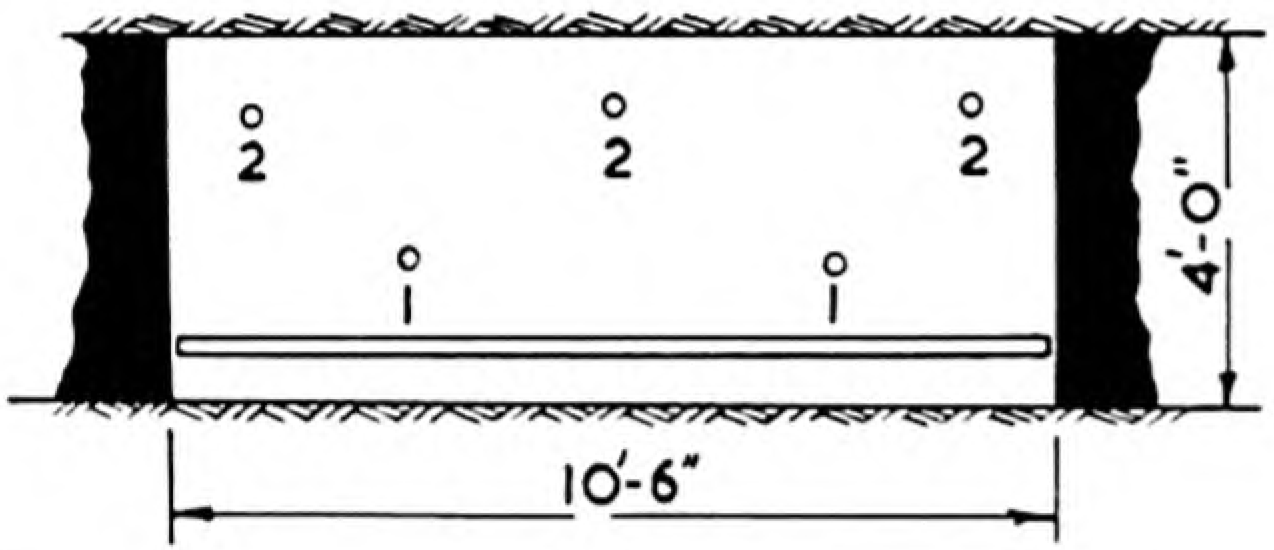


FIG. 223

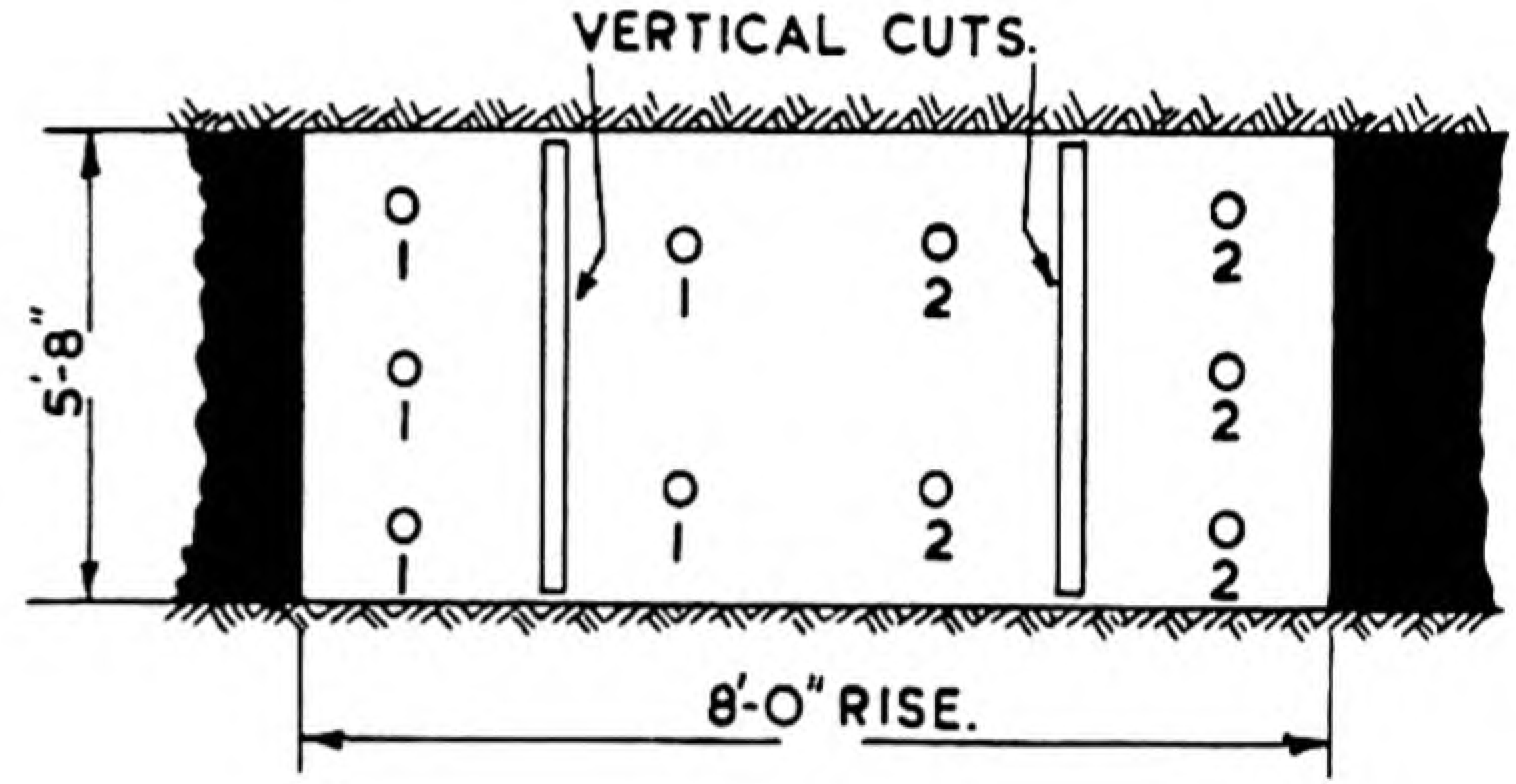


FIG. 224

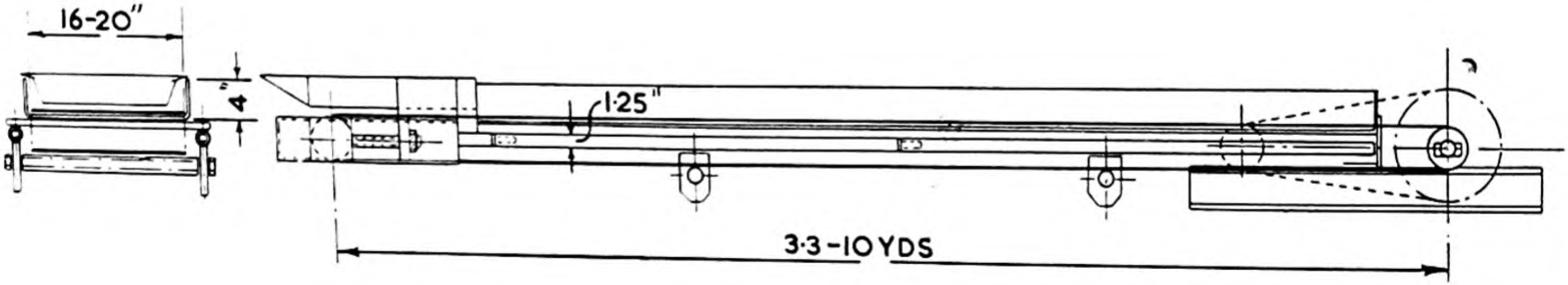


FIG. 225

filled, thus allowing the coal to slide evenly down the chute.

The bucket-scraper conveyor, like the belt conveyor, has the advantage of being able to haul in both directions, and therefore can be used for materials transport as well. The bucket-scraper can be used at flat and steep inclinations up to about 40 degrees, and can haul up to a length of 400 yards. At still greater lengths of drive, two scrapers can be used in tandem to increase the tonnage hauled, *vide* Fig. 226. The bucket-scraper delivery or loading-point midway up the rise is important. Both buckets must haul in different

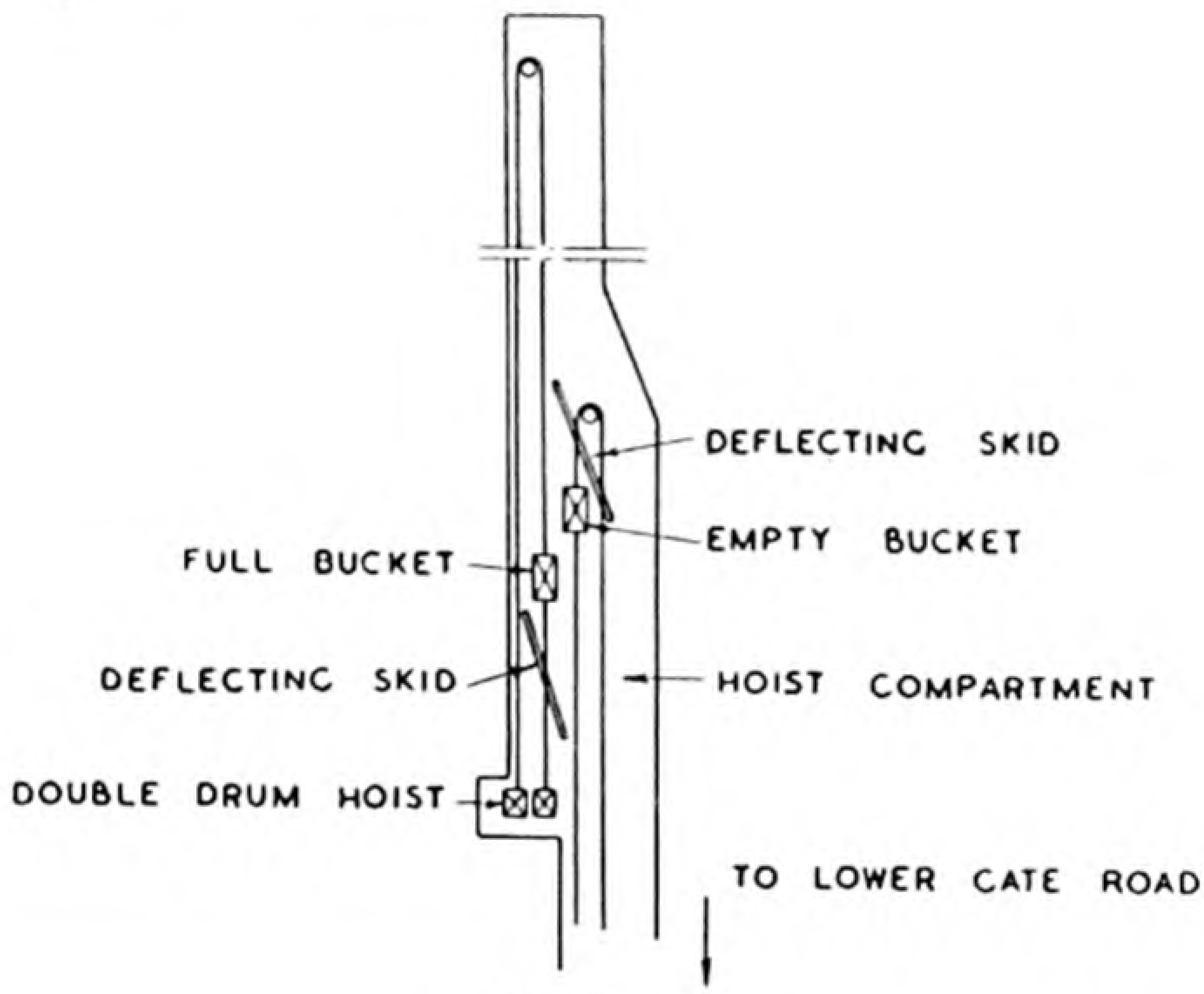


FIG. 226

tracks in order to deliver from one to the next outbye. The face-side bucket, or scraper-skip, is loosened from the two ropes in its own track at the delivery-point and fixed to the two ropes on the other outgoing track from the outbye scraper-bucket. The empty scraper-bucket is attached to the face-side bucket ropes and travels inbye for the next haul. The reversal of the buckets from one track to the other is done by guide rails which deflect the buckets into the other track. The arrangement is shown in Fig. 226, and the normal layout for a scraper haulage is shown in Fig. 227. A double-drum hoist or two single-drum hoists may be used, though the former is preferable. A 12-h.p. hoist usually would be adequate for the work, either with electric or compressed-air drive. The guide pulley is fixed by

suitable spragging between the roof and floor at the face end of the scraper haul.

The buckets, or skips, have a normal capacity of between 0·3 and 0·5 ton, and are long and narrow in construction. As an example, one skip was 6 feet long, 2 feet 6 inches wide and 1 foot 8 inches

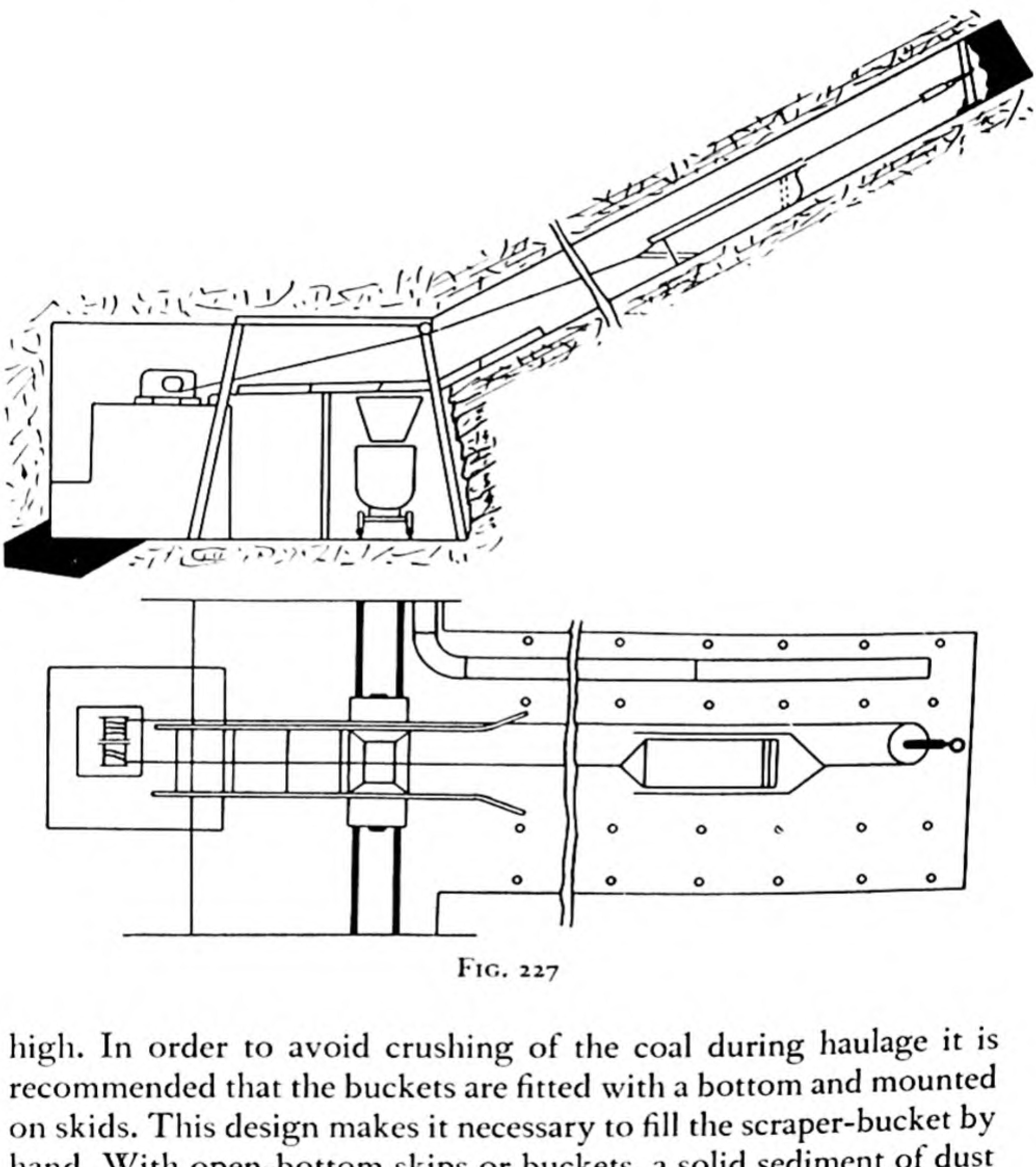


FIG. 227

high. In order to avoid crushing of the coal during haulage it is recommended that the buckets are fitted with a bottom and mounted on skids. This design makes it necessary to fill the scraper-bucket by hand. With open-bottom skips or buckets, a solid sediment of dust and small coal is deposited on the floor of the track. This dust accumulates as the scraper operates. In order to avoid this, the skip is fitted with a plough, which scrapes away this hardened bed of fine coal.

The side skids or lateral guides must be incorporated on the skip to prevent possible damage to the supports. The length of the skip should be at least double the distance between each row of props, but guide rails should be fitted to the bucket in most cases and bent towards the bucket interior in both hauling directions, *vide* Fig. 228.

In spite of the criticism of intermittent running and the necessity to have a wider rise (by about 4 feet), bucket-scraper haulage is used on the Continent in preference to belt- or shaker-conveyor haulage.

It is not true to say that the scraper haulage will hinder work at

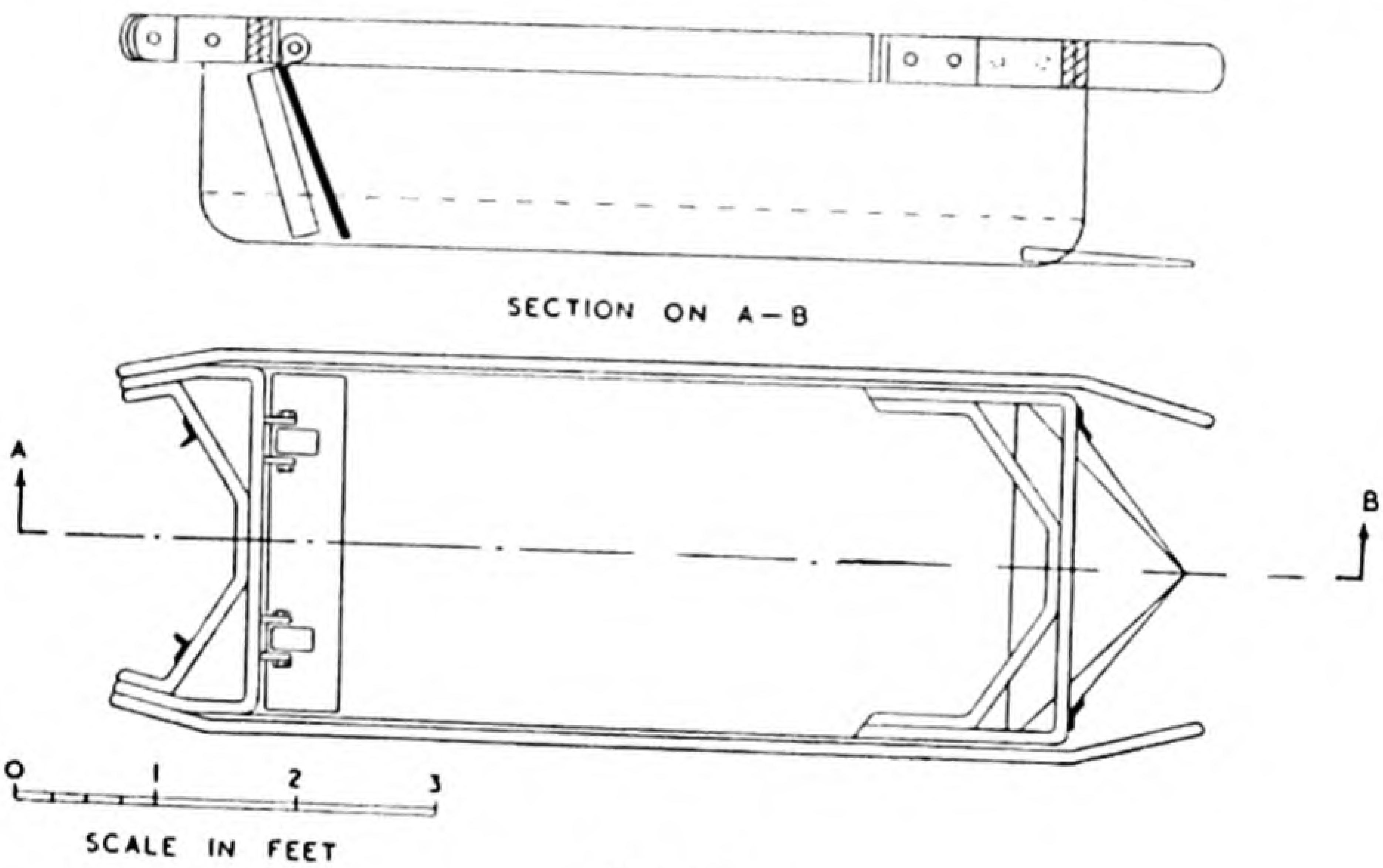


Fig. 228

the face any more than another type, even in narrower rises. This form of conveyor has also the advantage of two-way operation, and the running cost is only one-third of shaker haulage. This lower running cost is not based on the low cost of construction, since this is off-set by the high rope maintenance, but on a power cost comparison. The scraper, since it runs intermittently, is in use only when required, whereas the shaker or belt conveyor normally runs continuously. The scraper-bucket conveyor can be easily arranged to traverse small faults or uplifts of the floor, and since no additional work is required, as with shaker and belt conveyors, many Continental mines use this type exclusively for such duties.

(*c*) *Materials transport.* Shaker haulage needs an alternative

means of transporting materials, and supply sledges or special mine cars can be used. Fig. 230 shows several cross-sections of steep rises provided with a manway compartment. The sledge or mine car can be run in the manway compartment. The sledges are constructed in the form of a long, low car about 9 feet in length and 1 foot 6 inches to 2 feet wide, mounted on skids. The sledge is also provided with lateral guide rails bent into the front to prevent the carriage gripping the supports and ripping them out. A double-drum hoist is used to haul the carriage up the rise, the hoist being installed in the gate road. Two single-drum hoists also can be used, one in the gate road and one at the top of the rise. If at steeper inclination ladders are installed in the manway compartment, a sledge cannot be used and a car-type carriage running on track is installed.

The truck or materials car is constructed from steel plate and is about 12 feet long, 1 foot wide and 8 inches high. The height can be increased by using side plates set in vertical slots as in Fig. 229. The car is mounted on wheels the gauge of which corresponds to the breadth of the ladders, the main frame acting as the rails. The ladders are constructed of either wood or steel-rail with round iron rungs which serve as sleepers. The hoist used to haul the carriage is mounted at the top of the rise, the carriage being lowered by gravity. Since the position of the hoist and the necessity to move the hoist make it impracticable to install a large and heavy engine, it is sometimes necessary for a long haul to use a rope longer than can be accommodated on the drum. Lengthening ropes are used which are detached from the car as the car is wound up and attached again on the downward run. The car is held in the rise by a 'pronged bar' acting against a cross-strut when the additional rope is being attached or detached by the man accompanying the car.

A further simple method of procedure for materials transport in the rise can be applied where fixed chutes are in use. A winch is erected at the top of the rise and a direct rope haul used to carry up materials, using the fixed chute as a slide and fastening the material to the hoist rope. The rope is lowered by using a heavy chain as a weight. On the Continent, where the exploratory borehole system is applied, materials, such as timber, supports and compressed-air pipes, may be lowered to the face via the borehole, *vide* Fig. 189.

(*d*) *Supports.* Timber is normally used for support in the rise, since heavy steel props and straps are too heavy to handle. At least

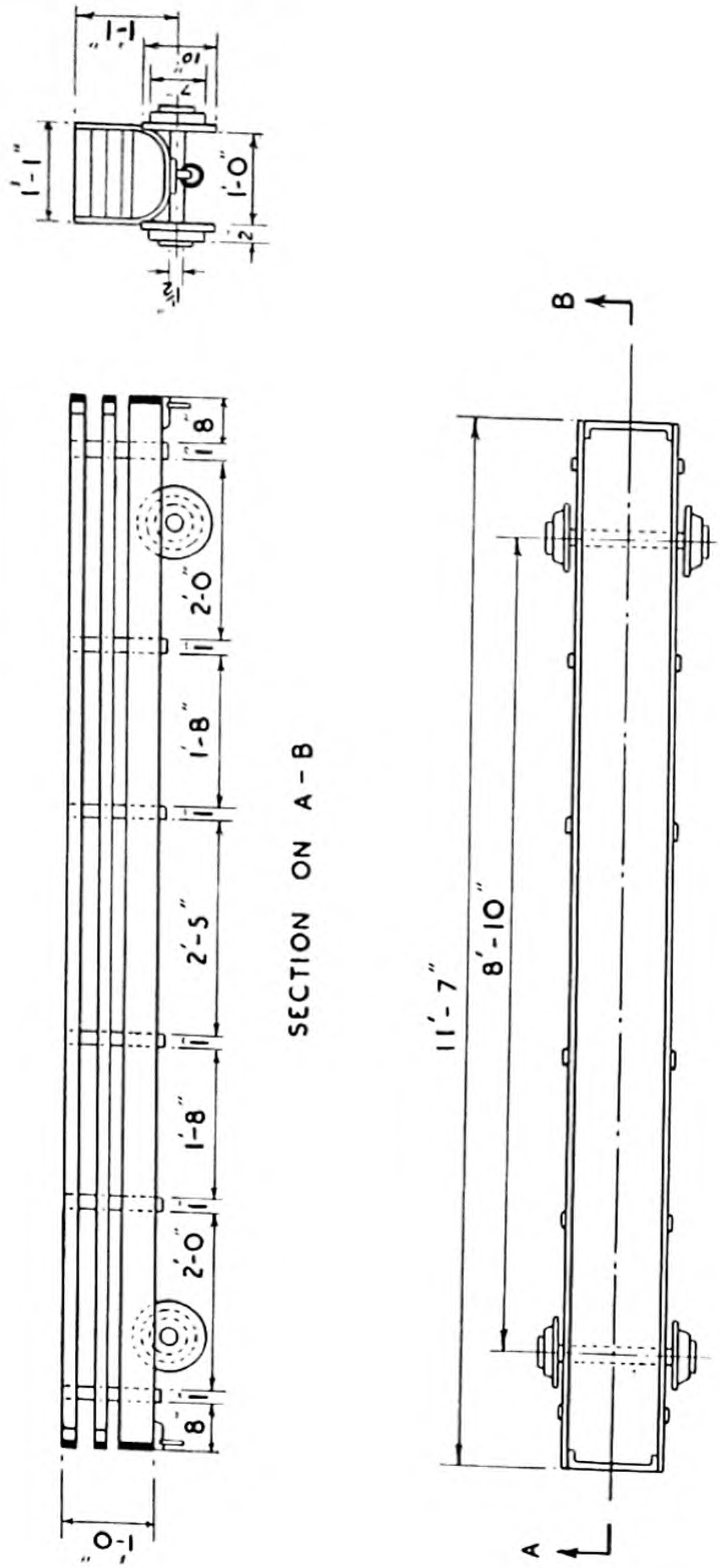


FIG. 229

two compartments are required, viz. a hoist-way and a ladder or manway. Three compartments may be necessary where the seam is less than 4 feet, since then there will be insufficient available space for air pipes and ventilation ducts in the working area. The supports should be set taking into account the provision for these services. Using cap-pieces 6 feet 6 inches or 10 feet in length, supported by either three or four props, an inside width of 3 feet can be arranged in each compartment.

The centre props should be set up to from 4 to 5 yards from the face so that the face men are not impeded by the timber. With seams up to 5 feet thick, props 4 inches in diameter will be sufficient; with thicker seams the props should be between 5 and 6 inches in diameter. Split props can be used as caps, since in the rises they are not required to take a high load. According to the quality of the roof material, close lagging may or may not be required, while at gradients over 25 degrees, the outside props should be tied by inserting round timber distance-pieces. The sets are installed from 3 feet to 4 feet 6 inches apart under normal conditions, or closer together, say 2 feet to 2 feet 6 inches apart, where the strata are friable.

If the rise is driven at a greater width than from 12 to 14 feet, for stowing waste or to provide more room at the working face, usually the additional space would not be supported by the cap-pieces. The space can be separately timbered or stowed as a side-pack. Some examples of the form of timbering used in rises are shown in Fig. 230. When deciding the location of the compartments for haulage, ladder-way, air and ventilation tubes, a number of factors must be considered. With a two-compartment rise, the haulage compartment may be set on the side nearest to the future working face. At lower seam inclinations the position of the haulage way, or hoist compartment, may be decided by the presence of pronounced coal cleavage. If the cleat is strong, the face will be set diagonally on to the cleat, as shown in Fig. 231. In this case, in order to keep the haulage close to the face and shorten the casting length, the compartment is set on that side farthest in advance. Where the seam is steeply inclined, the compartment must be set on the opposite side to that described, the coal from the face being deflected down on to the haulage by installing a wooden partition parallel to the diagonal face, as shown in Fig. 231.

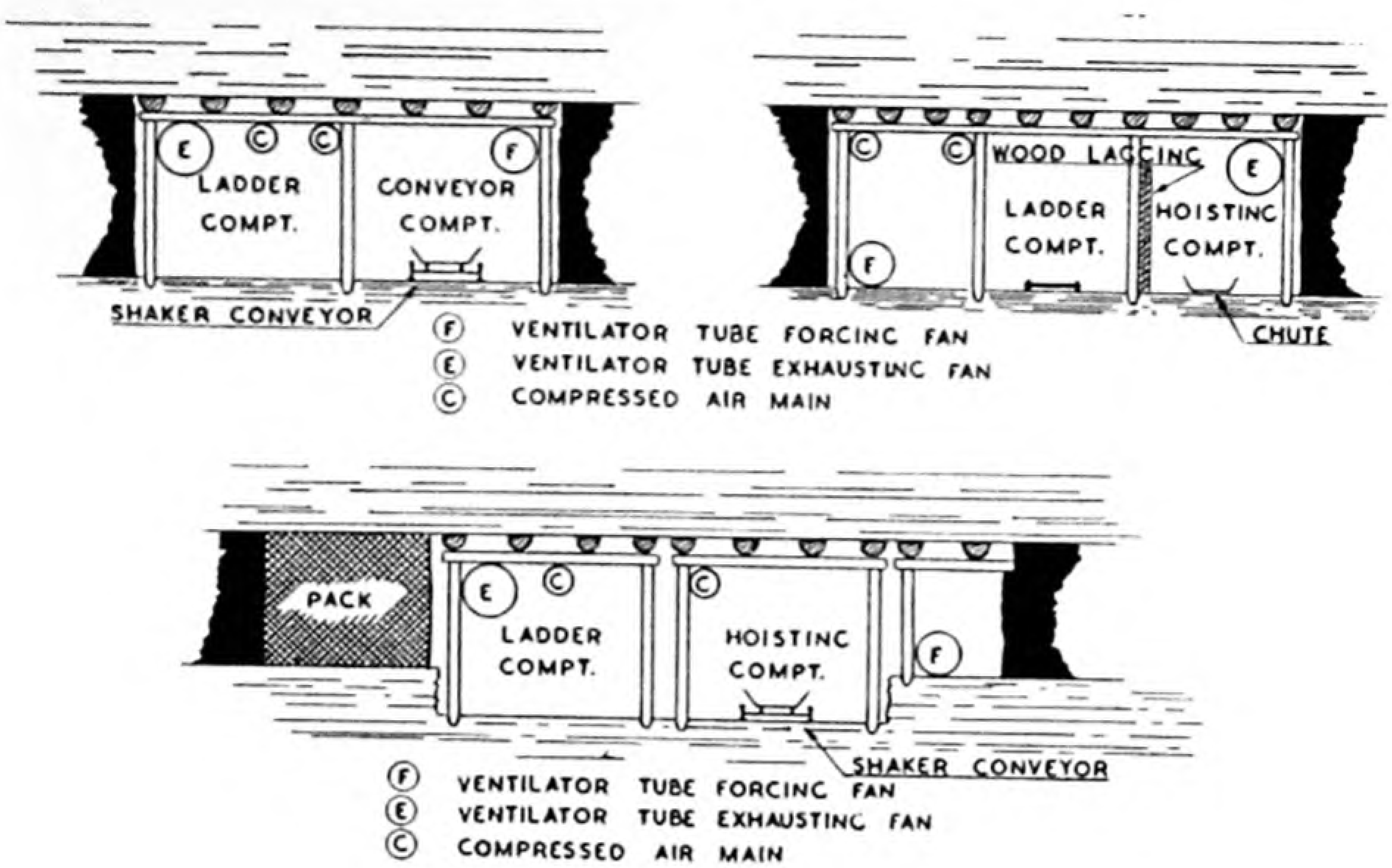


FIG. 230

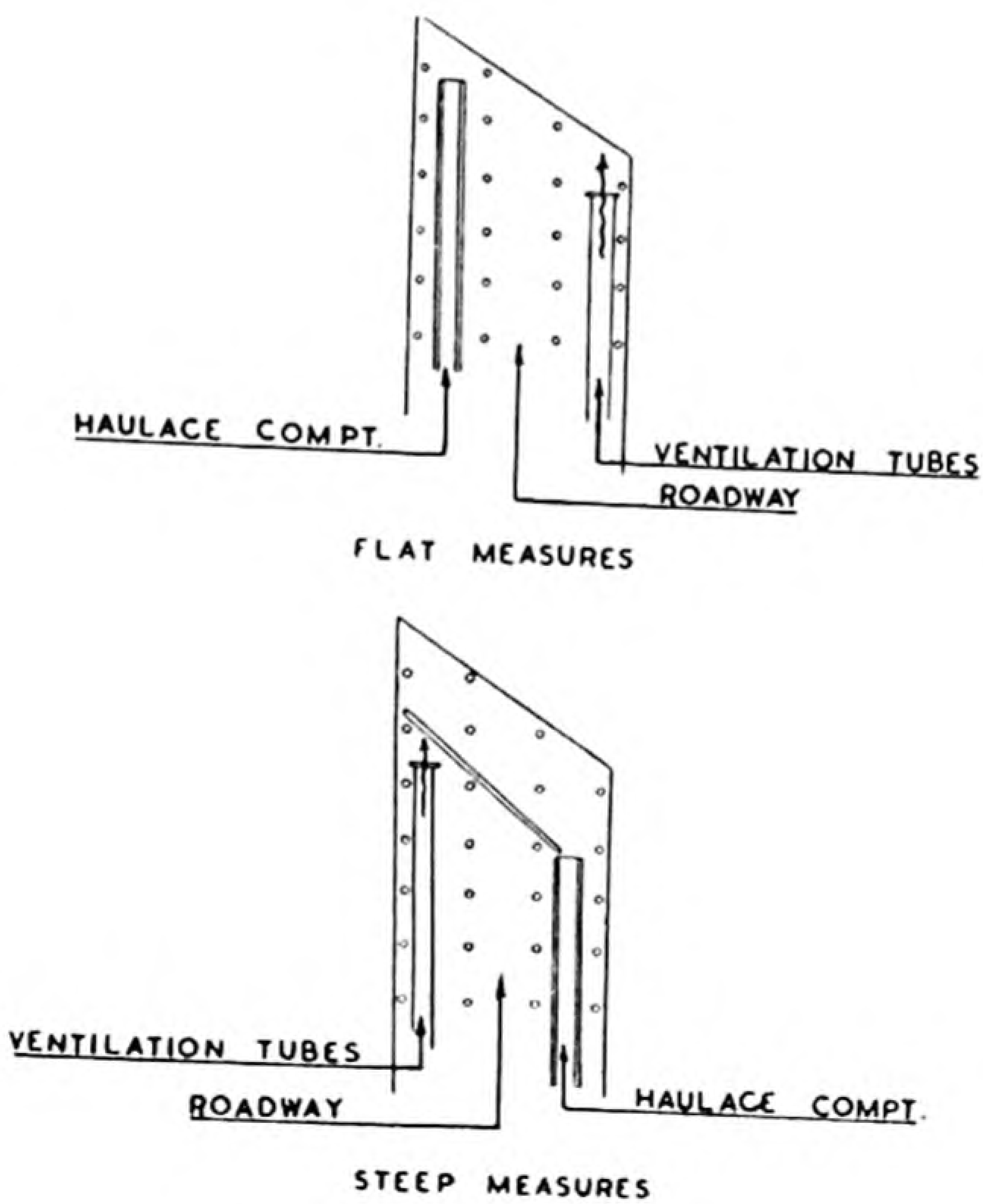


FIG. 231

In the case of a three-compartment rise, the factors to be considered are identical with those previously described for the location of the haulage compartment. The ladder-way is always in the centre, in order to allow easy inspection of the other two compartments. The ventilation compartment can be smaller than the other compartments. This is generally the case where scraper haulage is used, as the haulage compartment needs to be wider than normal. Up to inclinations of 25 degrees, the manway compartment requires no special equipment, but where the inclination is steeper, ladders should be installed.

(*e*) *Ventilation.* Special attention must be given to the provision of adequate ventilation, since the rise is not on circuit with the main ventilation system. An auxiliary fan must be used. In a non-gassy seam, an exhaust fan can be used, in addition to which a compressed-air nozzle at the face in a short 'Venturi' tube keeps the air in circulation at the face.

In gassy seams, two separate and complete ventilation tube installations are recommended. The forcing fan tubes should be installed nearer to the face than the exhaust line, the exhausting fan and range being larger than the forcing side. The quantity of air required is generally from 1,200 to 2,000 cubic feet per min.

With steel air ducts the flange type is preferred to the spigot type, as leakage is less liable to occur. Canvas ducting, which can be advanced quickly, may be used with forcing fan installations.

The number of fans required will depend upon the efficient installation of the ducts, but generally one fan is used per 400-feet length of air duct. The duct diameter is usually 20 inches, though in thin seams this may be 16 inches.

A further means of assisting the ventilation in gassy mines, when driving to the rise, is the drilling of a hole in advance of the heading. The hole (rat hole) drilled in the coal is from 8–15 inches in diameter. The holes are drilled, using a special rotary boring rig, boring being done from the lower to the upper level. The drilling performance varies according to the hardness of the coal and may be up to 150 feet per shift. A major disadvantage of the method is the great deviation of the hole. The deviation can be lateral or, where dirt bands are present in the seam, the hole may be deflected towards the floor. For this reason, with holes longer than 300 feet it is difficult to keep the deviation within the limit of the rise section.

The bore hole has the decided advantage of providing a through ventilation to the rise, which is particularly important where the gas emission is high. The improvement of the ventilation will allow the use of delay-action firing and the cost of auxiliary fan ventilation is reduced or eliminated. The forward boring has the advantage of proving the ground, especially in the case of the possible presence of old workings.

A speaking-tube and audible bell signal should be installed to assist in the efficient running of the work, and both of these can be placed in the ventilation compartment. Good lighting at the face is also required, and for this, air turbo-electric lamps or large accumulator lamps can be used, though, if it is possible, mains lighting is preferable.

(*f*) *Organisation and performance.* The number of shifts worked in the rise will be two, three or four per day according to the necessity for a slower or more rapid rate of advance. In every case, it is necessary to organise the work so that a regular rate of advance is attained per shift. If a special hoisting system is required for the materials transport and additional men have to be employed (usually when the rise is longer than 50 yards), the development cost for a three- or four-shift cycle is less than for a two-shift system; the same is true where a belt conveyor is used, and requires to be lengthened by additional men. Because the additional wages cost is distributed over three or four shifts instead of two, the development cost per yard advance is lower.

The smallest number of men per shift will be two, though often three will be required in flat seams, where casting has to be done, or in steep seams due to the more difficult timbering. With very hard coal, and where the face is more than 220 square feet, a fourth or fifth man may be desirable for materials transport and other work. In addition to the face men, a loader man is required at the lower level, either to operate the loader point or the scraper hoist.

When fixing the contract for the work, payment should be based on the advance or yardage and on the number of tubs filled. Where the rise is paid on yardage rate only, there is a danger of the rise being driven at a width less than specified; with the payment system based on a tub price only, the rise may be driven too wide.

The performance per manshift, exclusive of the loader and addi-

tional transport men, can vary between 20 and 50 inches, depending upon the seam conditions and the method of coal-getting used. With a three-shift system and three men per shift, the heading will advance at from 7 to 10 yards per day. With intensified working under favourable conditions, a rate of from 12 to 15 yards per day may be achieved.

Section 3. Driving Rises to the Dip

The method of drivage and the system of timbering adopted in dip-rises does not differ from the normal procedure where the heading is driven to the rise. The haulage or hoisting becomes even more difficult with increasing inclination. Belt and chain scraper conveyors are recommended where the inclination is not great, the belt conveyor being used up to 16 degrees and the chain scraper conveyor up to 35 degrees. In every case with increasing inclination, the hand-filling at the face becomes more difficult and the bucket-scraper, which is the usual means above 35 degrees, will frequently be preferred throughout. The scraper-skip or bucket is usually a box construction and runs on skids. The ventilation is normally by means of a forcing fan or with the advance bore-hole previously described. Where the seam inclination is at least 60 degrees and the coal is mainly smalls, the bore hole can be used for transporting the coal to the lower level, in which case the advance is increased enormously. Since the heading is to the dip, the concentration of gas at the face is very slow and shot firing may be carried out, using delay-action firing where this can be adopted. This advantage, however, does not outweigh the disadvantage of dip-drivage, and rises driven up the seam gradient are generally preferred.

Rises driven as shortwall faces. In the same way, gate roads can be driven in flat inclinations by adopting a short face system of development, and two rises are driven, the face between the rises forming a short panel 10, 20 or even more than 30 yards long. The advantage of this method of development lies in the provision of 'through' ventilation and its effect on the performance in yards per shift and in the reduction of the overall cost. The disadvantage is, however, in the slower rate of advance, which may be only from 4 to 6 feet per day, and in the greater proportion of time for the preparation of the winning face. The individual cycle of operations, including drilling, firing, haulage and stowage, are the same as in normal face develop-

ment. The stowing material is ripped from the rise headings or using dummy roads on the face and strip packing. The main rise can be in the centre or at one side of the panel, but is usually central, as illustrated in Fig. 232, with small rib-side gates for ventilation purposes left in the side-packs. Where the rise heading is at one side, only one rib-side road is necessary on the other side, *vide* Fig. 208. The rib-side road forms the winning face of the future panels.

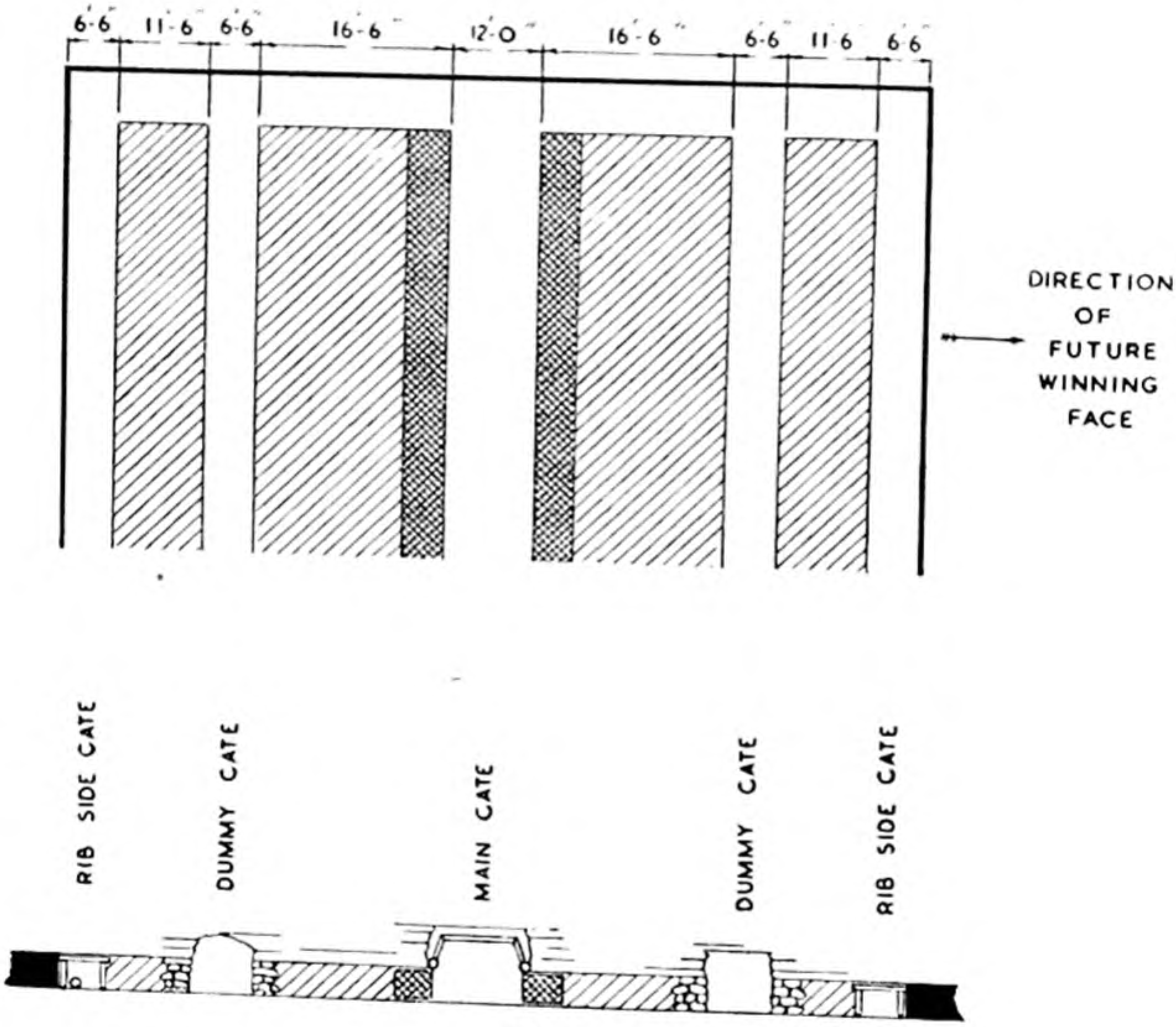


FIG. 232

In British practice, where flat measures prevail, face development by shortwall panels is frequently applied. On the Continent, however, the single-rise heading is preferred in most areas, the shortwall-rise development being confined to special cases. This may occur where the rise is being driven alongside a fault. With a small heading, the winning face development would be difficult, whereas a short face can be lengthened or decreased without difficulty as the direction of the fault changes.

CHAPTER 5
TRANSPORT

PART I

TRANSPORT IN GATE ROADS

Section 1. *Conveyor Transport*

(*a*) *Rubber belt conveyors.* The rubber belt conveyor consists essentially of a gear head or driving unit through which the belt is laced in several ways, the belt being supported on a structure carrying idler rollers taking the top and bottom belts and a tension or return end-assembly.

The belt is composed of woven cotton layers (plies), impregnated and covered with rubber and then vulcanised. The use of rubber belts has increased the fire risk and recently non-inflammable belts have been introduced, on the Continent with the rubber displaced by neoprene and in Britain with a polyvenylchloride base.

The parallel-faced driving drums are made of steel and keyed to the steel driving shaft through cast-iron bosses spigoted into the machined steel end-plates and dowelled. The drums are carried in ball bearings mounted in housings spigoted into the side-plates of the head structure, the bearings being fully protected by oil seals. The surface of the drum is generally smooth, but for use in wet conditions it may be grooved or lagged with wood or lined with rubber. The diameter is kept as large as possible in order to maintain the maximum grip on the belt, to reduce wear and assist in passing the belt joints over the largest possible drum diameter. The diameter may vary from 10 inches with small-powered gear heads to 24 inches with the largest, while in trunk conveyors the drum diameter may be as much as 42 inches. Limitations in the height of the unit necessitates the maximum use of space on the gear-head framework in order to keep the drums as large as possible. The belt is held in contact with the largest possible surface area of the driving drum by means of a jocky pulley, generally spring-loaded. There are many modifications in the assembly of the primary drive, jockey or mangle roller drums, either one or two driving pulleys being used, depend-

ing upon the power to be transmitted. Face conveyors usually have one driving drum and large trunk conveyors two. Sectional views of a typical gear head are shown in Fig. 233.

The drive from the gear box to the drum or drums is through a dog clutch or clutches. On some makes of gear head, a dog clutch is fitted to both ends of the driving drum shaft, so that, in case of breakage, the drum can be removed and reversed. The position of

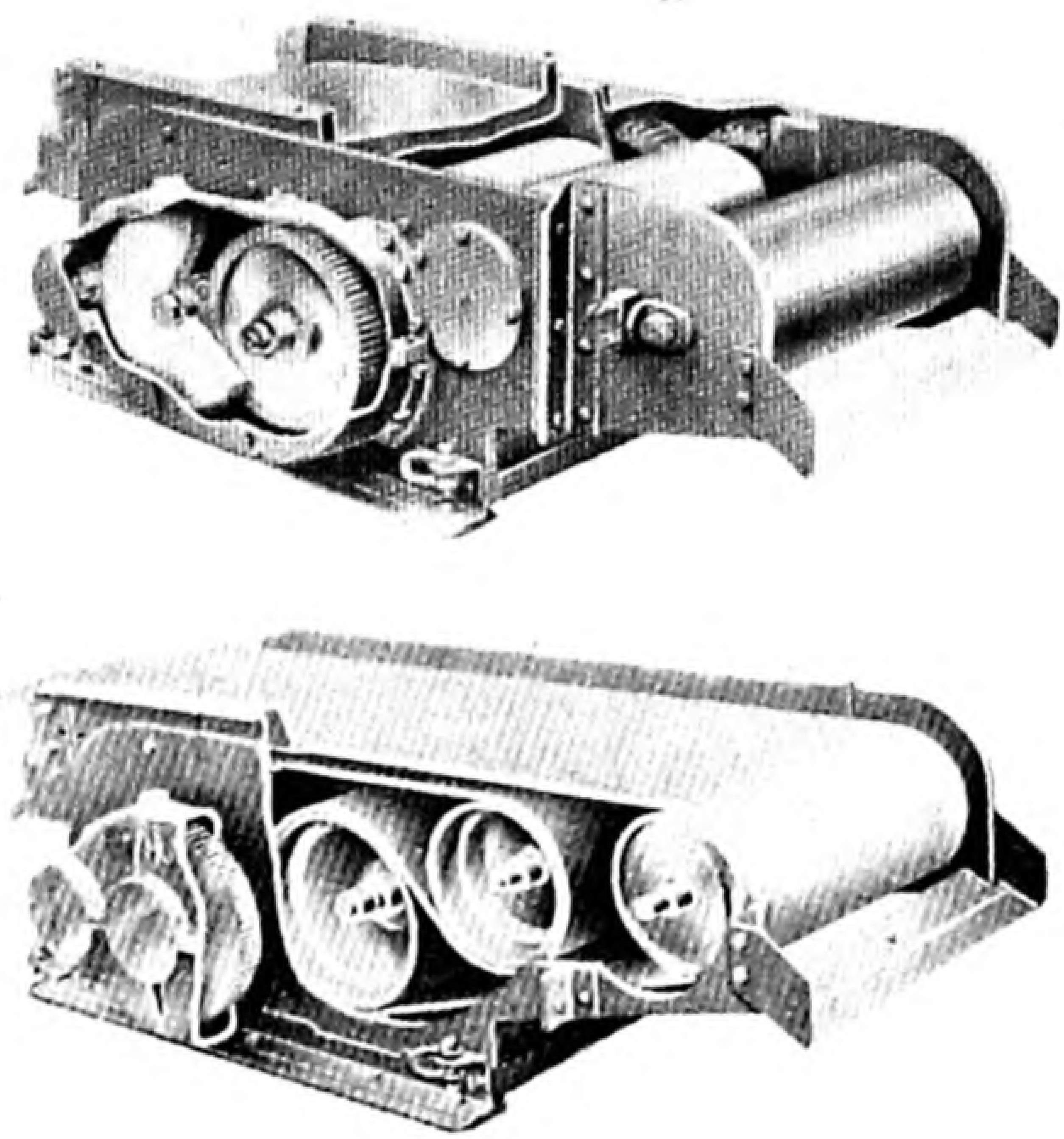


FIG. 233

the drum or drums, the belt lacing and the location of the pulleys in two types of gear head are illustrated in the diagrams in Fig. 234.

The gear box is generally mounted at the side of the head structure and it is usually possible to reverse the position to either side of the gear head. The gearing can be either a double reduction spur and pinion gear or single or double worm reduction, the latter type considerably reduces the noise in operation, while the gear ratios can be varied to give any desired belt speed from 150 to 450 feet per minute.

Most driving units above 15 h.p. are arranged to drive on the return belt, the two-drum assembly giving a belt wrap of about 500 degrees and providing a ratio of initial tension, when the belt

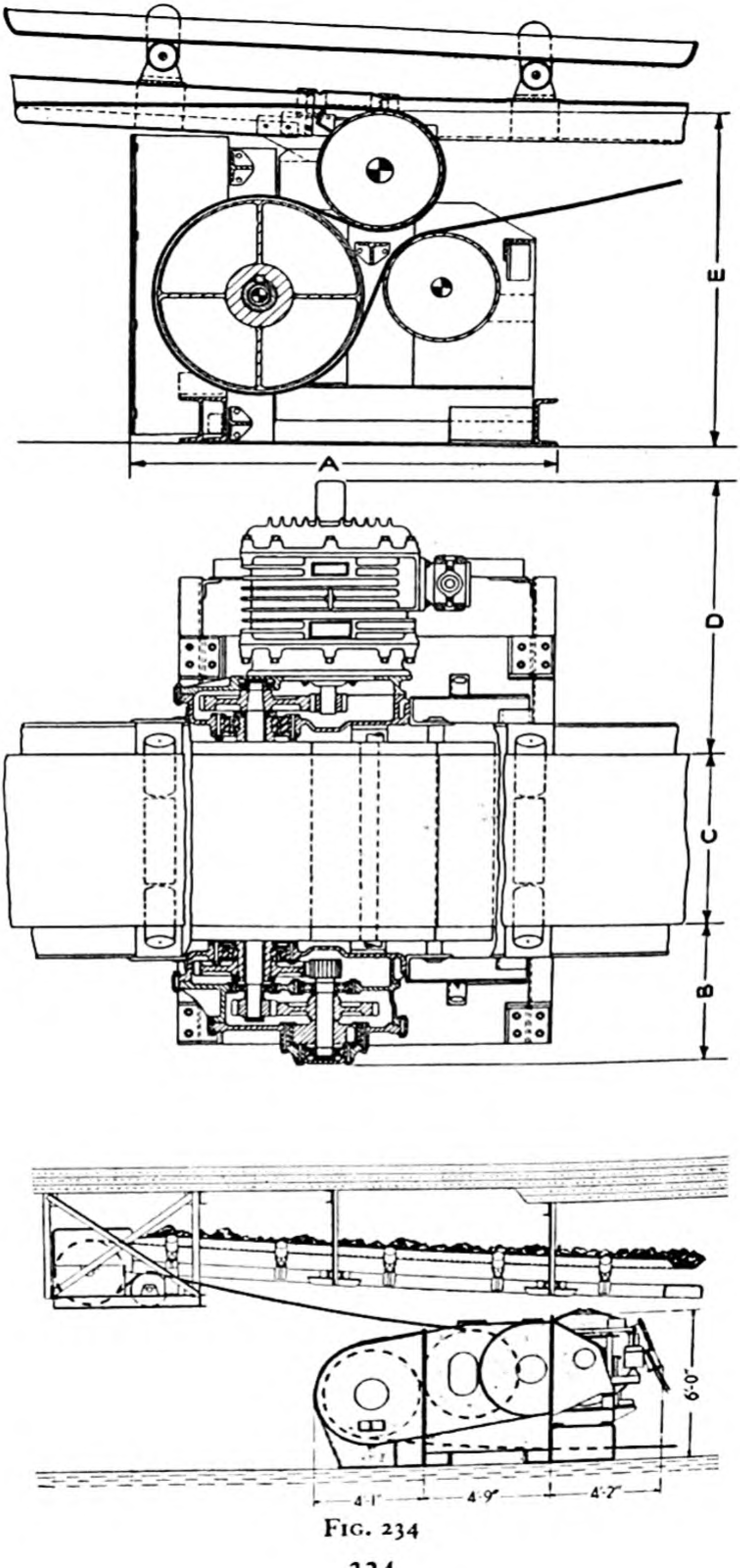
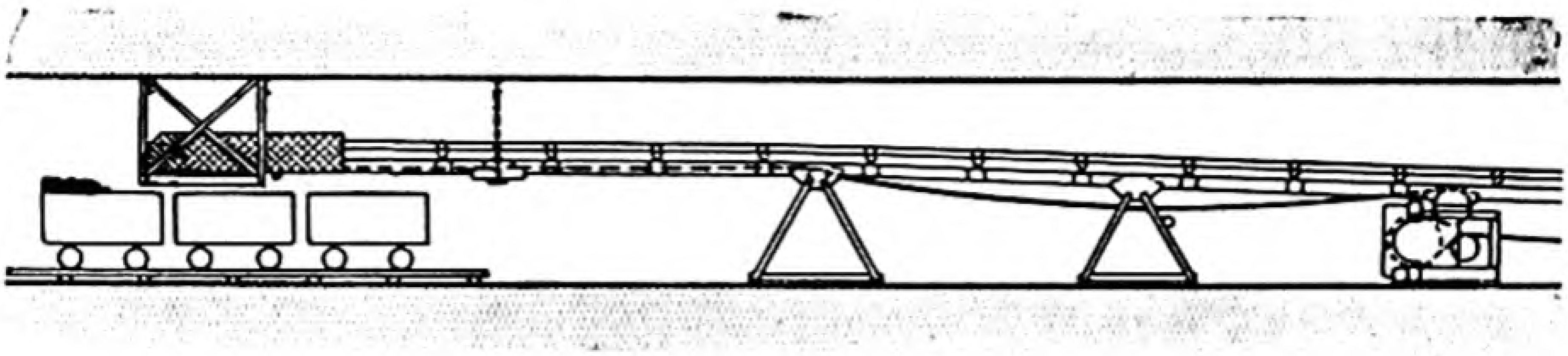


FIG. 234

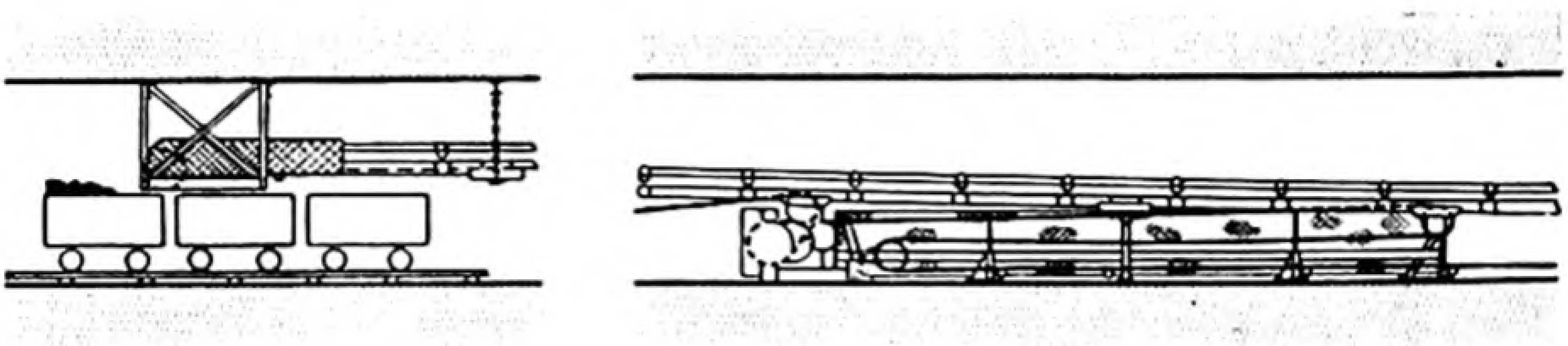
leaves the driving drum, to driving tension sufficient to drive the belt. If this ratio is taken as 4·3 and the gear has to exert a pull of 1,000 lb., the initial tension must be at least $\frac{1{,}000}{4\cdot 3}$ or 230 lb. The difference between tight and slack side tensions is then 770 lb., and this is the force available for driving the belt. In modern gate-conveying practice, the driving unit is usually placed on a prepared foundation on the floor and the delivery of the material over a remote delivery or boom which may be at a convenient distance from the driving unit. The driving unit is usually placed underneath the conveyor, when conveying on the level and driving on the bottom belt. This position keeps the gear away from the loading point and makes it convenient to install and maintain. The plain head pulley used at the delivery end is light and easy to install, since it is smaller than the driving drum and since it can deliver close to the top of the tubs, it decreases the drop of the coal and the consequent breakage, less height is also needed at the delivery point. The several diagrams shown in Fig. 235 illustrate the position of the gear head under varying conditions when conveying on the level, uphill and downhill, and also show the remote delivery boom.

Most gate conveyors employ a loop take-up system, making the daily extension of the conveyor much quicker and simpler. The loop usually stores 60 feet of belt, enough to extend the conveyor 30 feet, although longer reserves can be provided. The standard unit illustrated in Fig. 235 (b) and (d) comprises a fixed drum and a winch-operated sliding drum, mounted within a steel structure below the carrying belt and away from the delivery end. The belt running out of the driving unit passes inbye to the fixed drum, round which it runs outbye to the sliding drum, returning inbye in the normal return belt position. The winch drives spur gearing, which turns two rope or chain pulleys, hauls in the ropes or chains, and so pulls the sliding pulley along the guides. A pawl and ratchet retain the tension, and wire mesh guards on both sides of the take-up enclose the belt and all other moving parts. The loop take-up is also illustrated in Fig. 236, while a take-up winch is shown in Fig. 237 and the complete unit in Fig. 238.

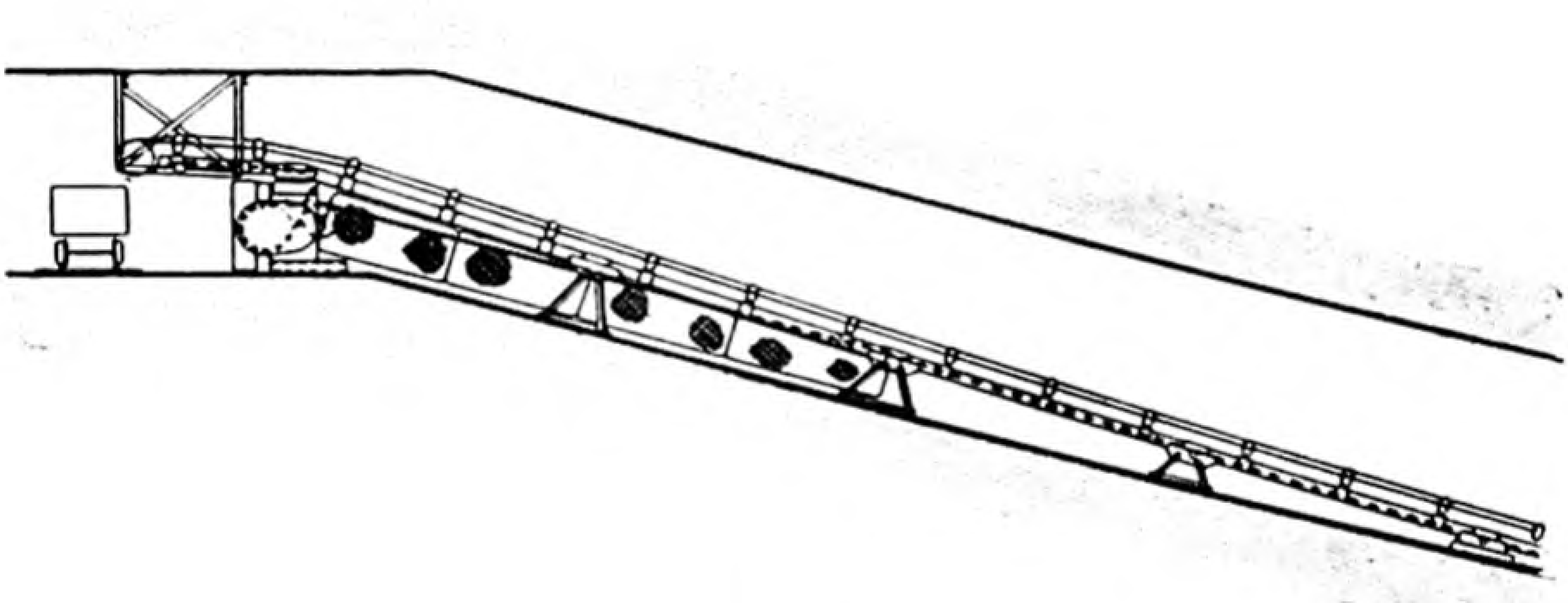
The belt-conveyor structure may be formed of inverted steel troughing, from 8 to 12 feet long. The troughs are strengthened



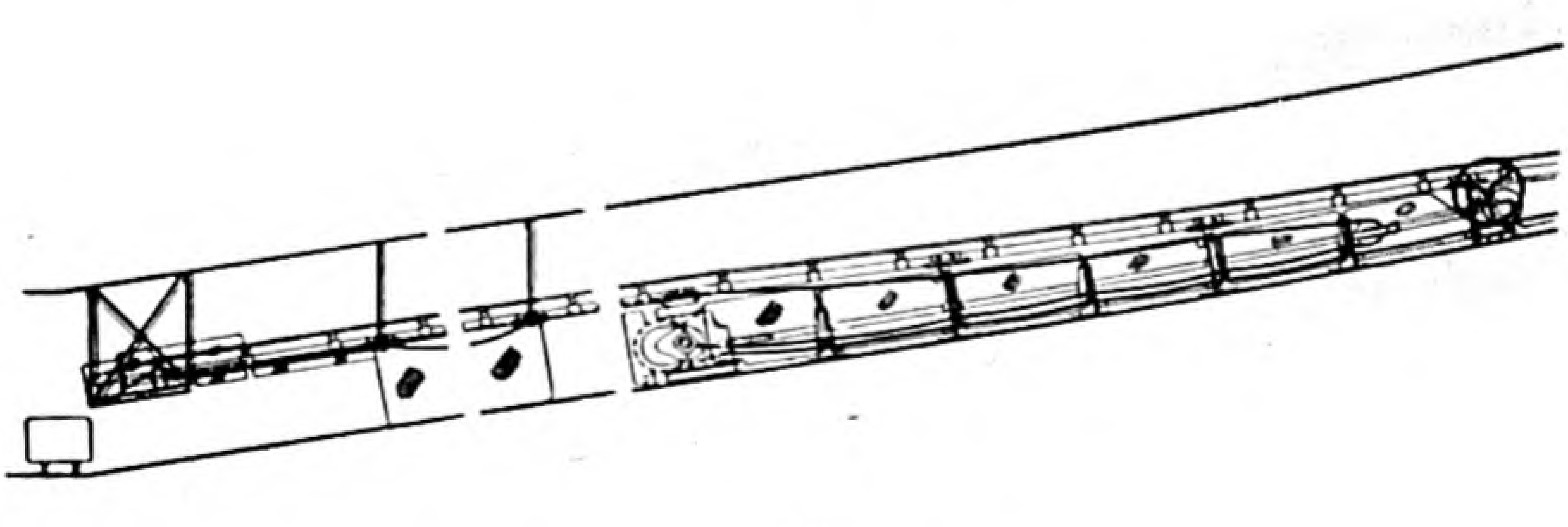
(a)



(b)



(c)
Conveying uphill.



(d)
Conveying downhill.

Fig. 235

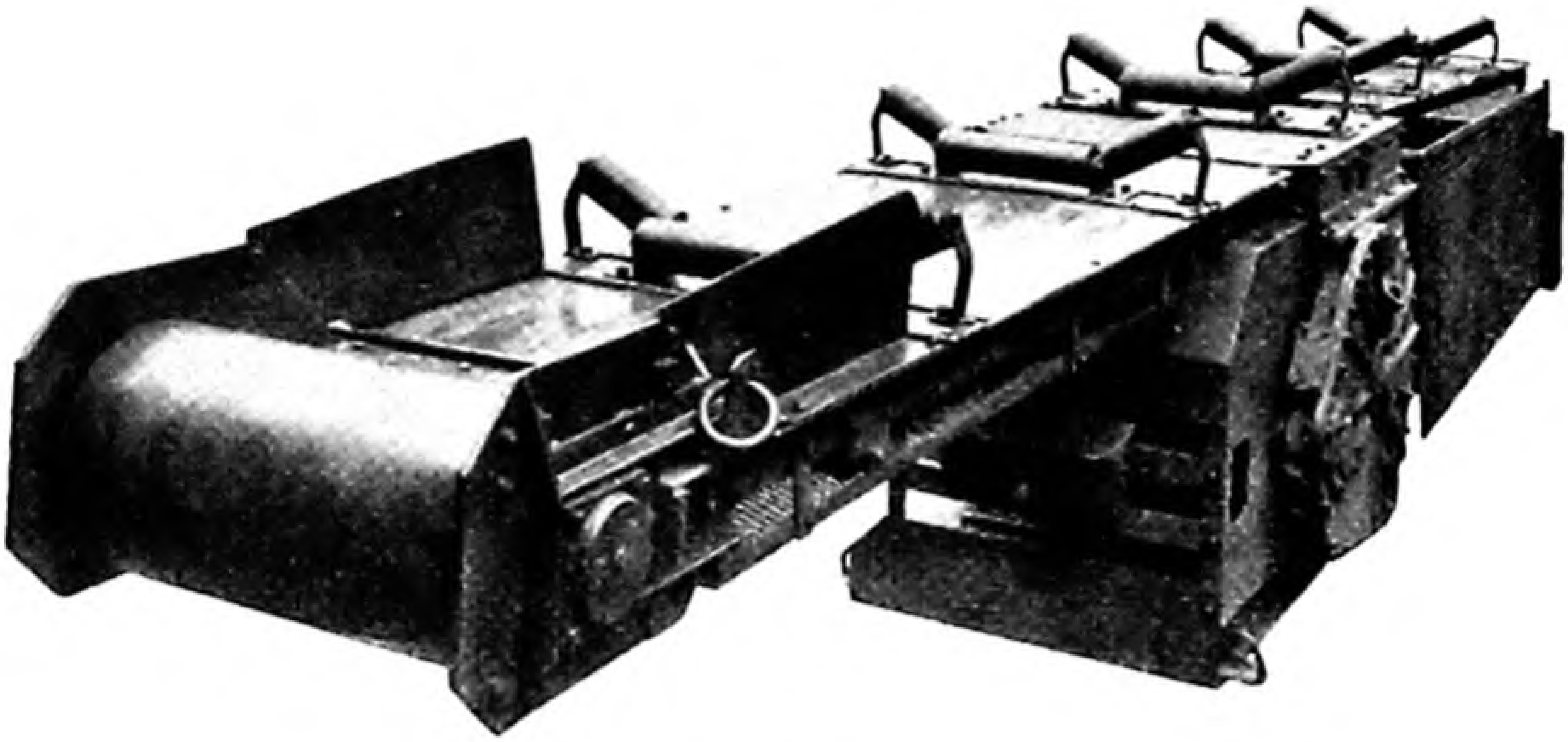


Fig. 236

by corrugations, by embossed saddles under the idler brackets and by reinforcement pads riveted on to provide seatings for the idler brackets. A typical section of M. & C. structure is also shown in Fig. 238. The Meco autolock structure accommodates 24-, 26- and 36-inch belts, and is easy to handle and quick to erect. The sections are

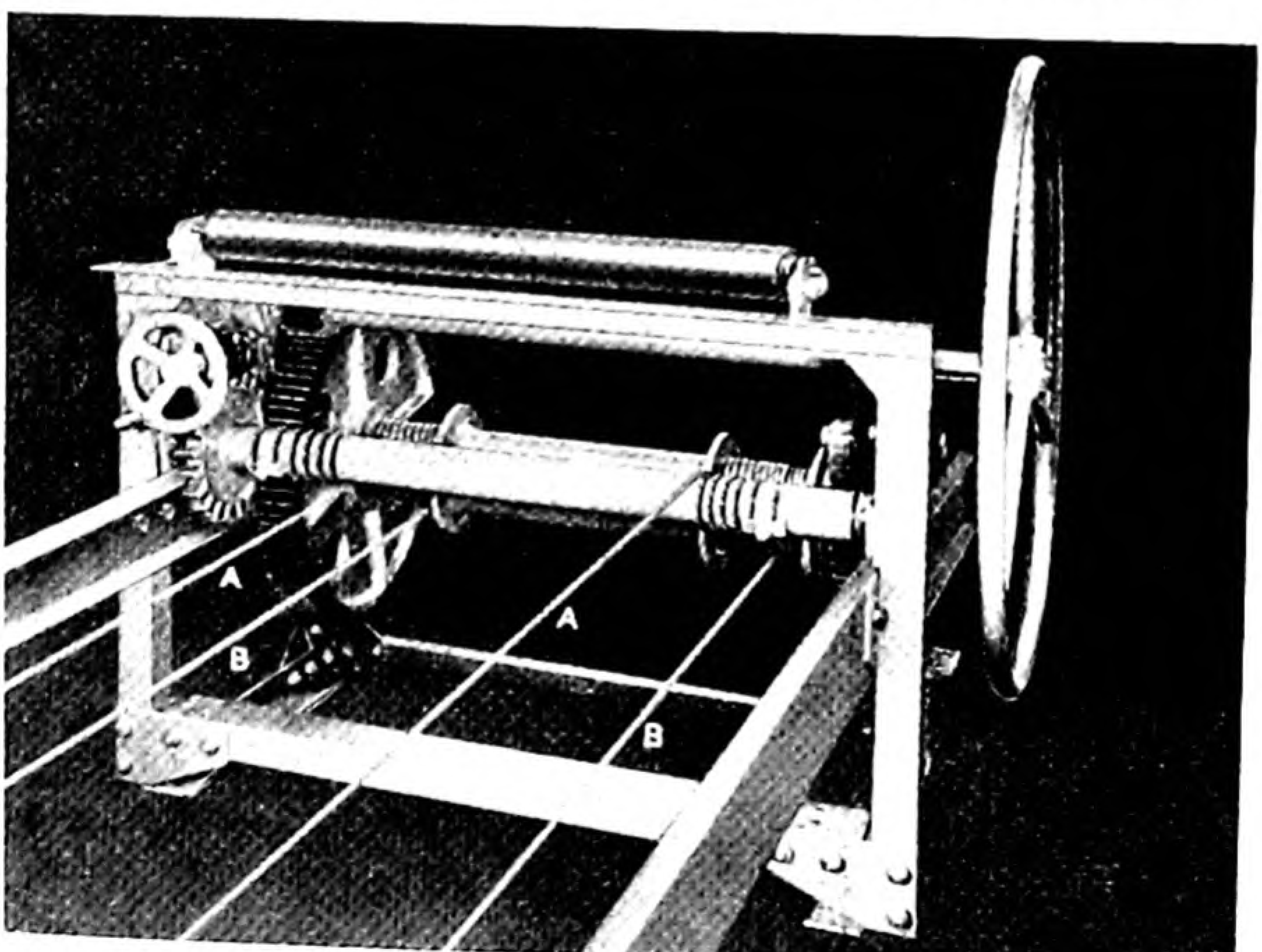


Fig. 237

9 feet long and are of pressed channel section with cover plates half the belt width, strengthened with riveted plates at each end. The troughed idlers are mounted on 'U' straps and fitted by means of two countersunk bolts, the half-covers locking automatically on the stools. A section is illustrated in Fig. 239, in which a three-roller idler is used, spaced at 4-foot 6-inch intervals. The M. & C. system is to employ standard idlers and brackets in a 'piecemeal'

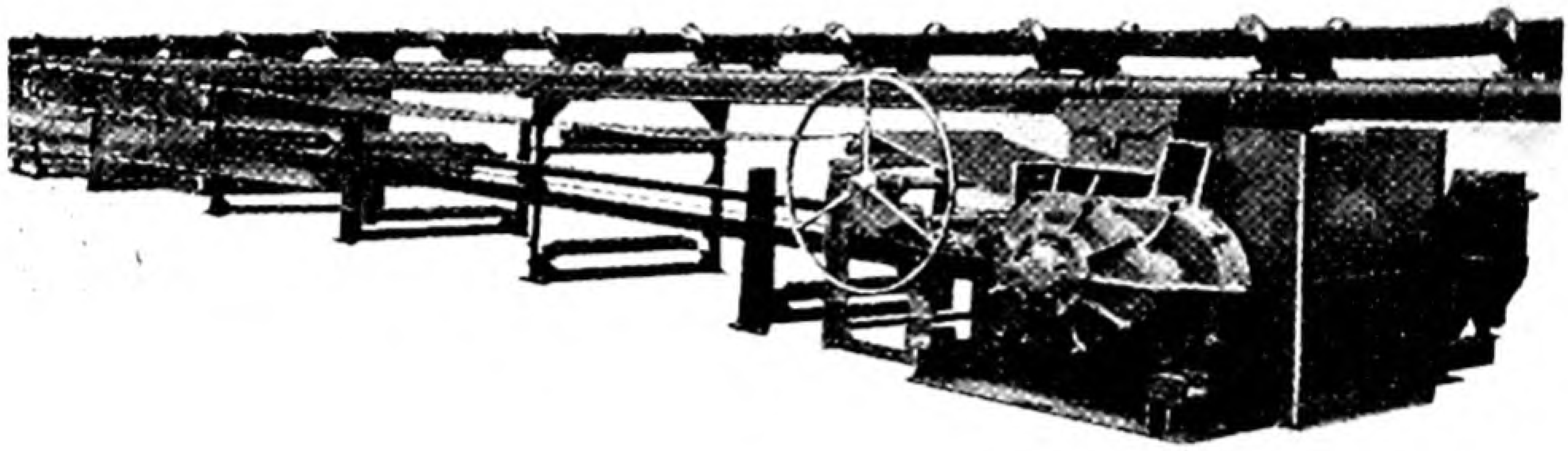


FIG. 238

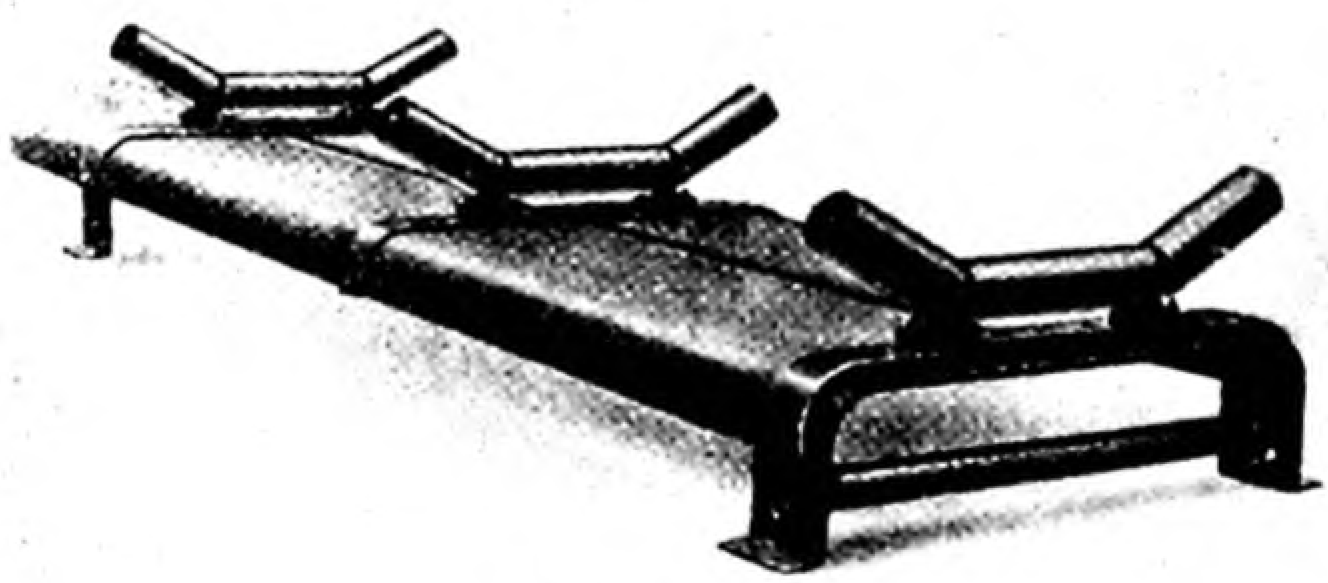


FIG. 239

structure fitted together without bolts or tools. In every case the bottom belt is completely protected by the top cover.

A particular form of stringer structure, as shown in Fig. 240, may be used to provide a conveyor structure which will resist floor movement and facilitate realignment. In one type the troughed idlers are mounted on channel-section structures bolted to continuous base angles of heavy section which are fish-plate jointed. Louvres extending inward over the return belt are sprung into position over pegs and angle cleats, and thus form with the base angles a rigid, strong structure capable of spanning bad ground and allowing packing up for alignment.

The return end-assembly as shown in Fig. 241 may be used to

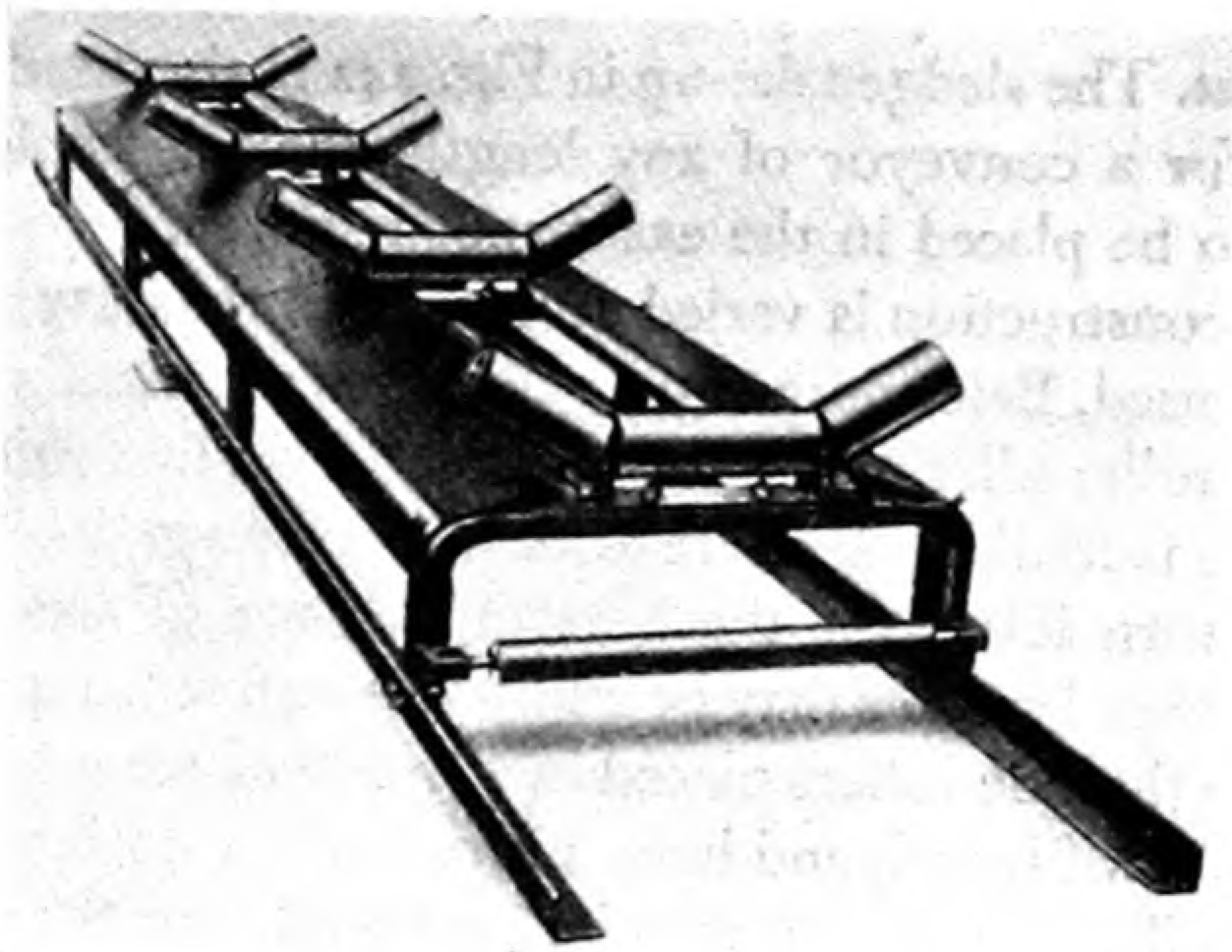


FIG. 240

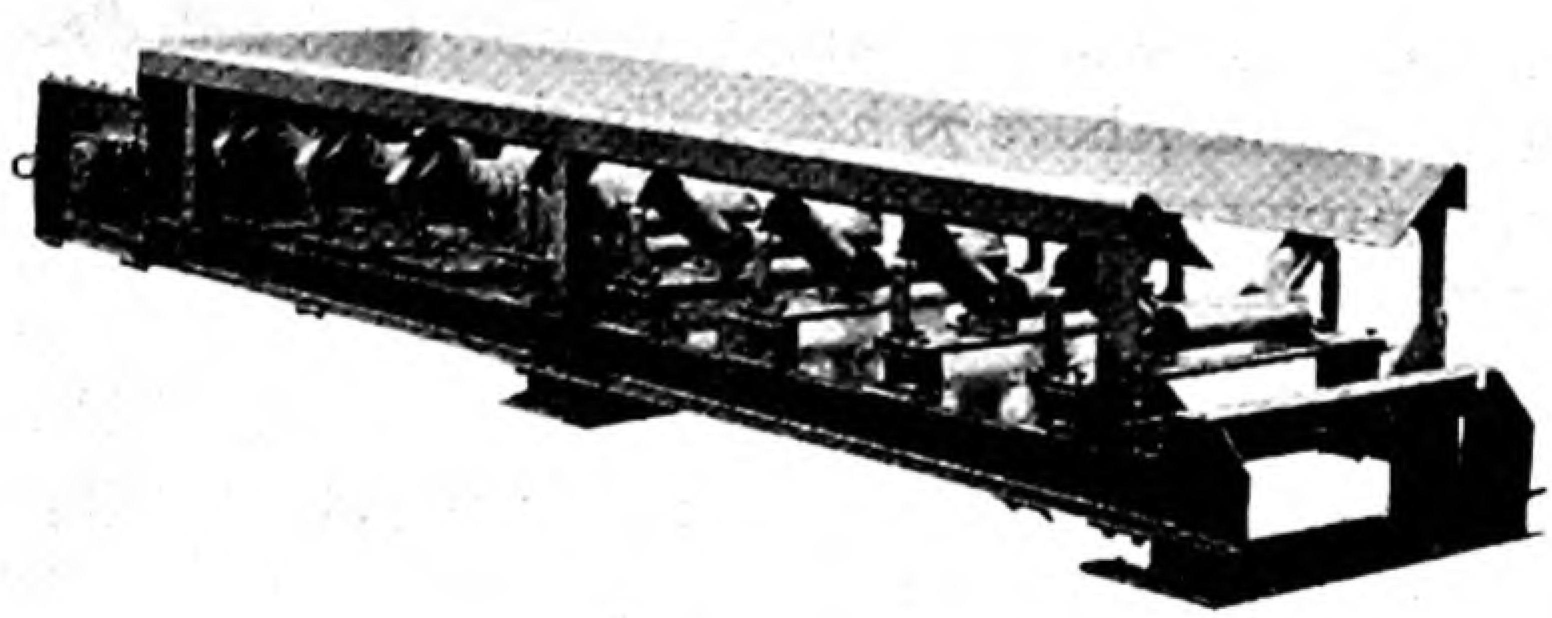


FIG. 241

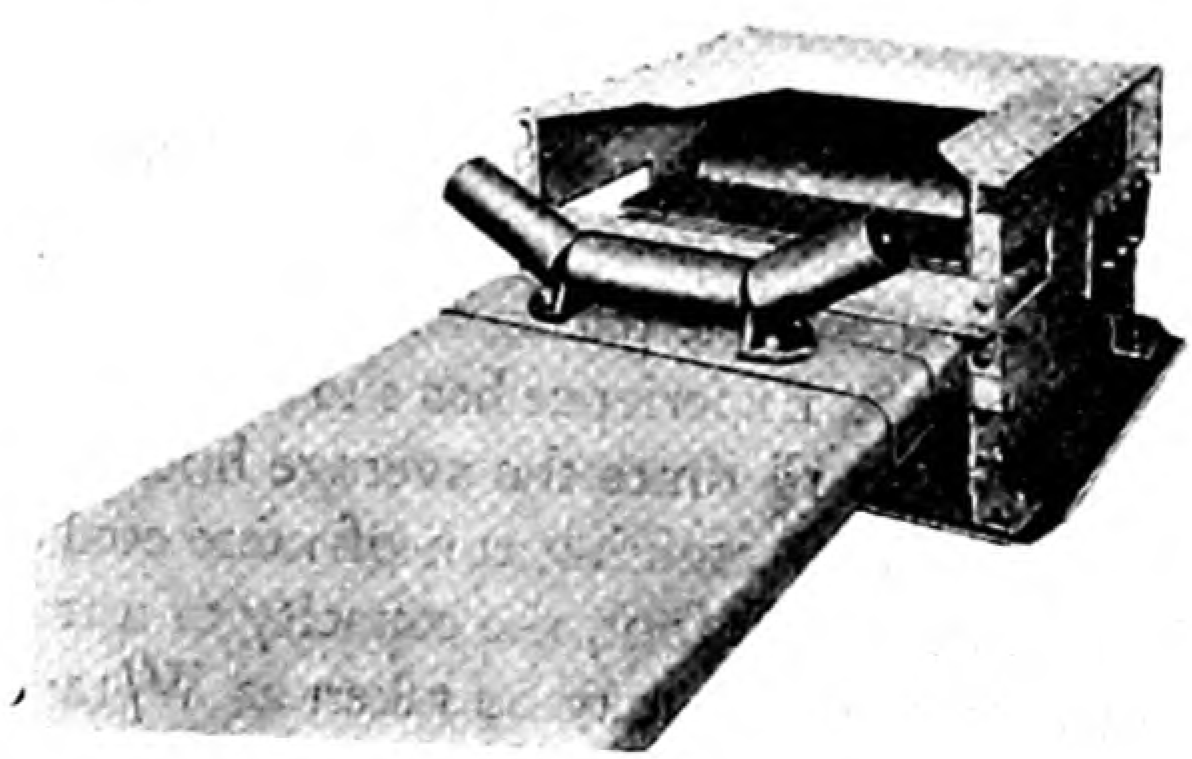


FIG. 242

apply tension if the tail pulley is not close to the face and has room for movement. The sledge take-up in Fig. 242 is also used as a fixed return end for a conveyor of any length, the telescopic troughs enabling it to be placed in the exact spot required.

The idler construction is varied according to the duty and width of belt to be used. Belt widths up to 36 inches are usually provided with a three-roller idler, while for 42-inch belts a five-roller idler is used. The top belt idler rollers are made from close-grained cast iron, while the return idlers for the bottom belt are of seamless steel tubing. The idler brackets may be adjustable or fixed and the rollers set in line, or the side rollers placed in advance of the middle roller, in the direction of travel, and have a forward 'set' to centralise and steer the belt. The roller hubs may be grease-filled or the rollers oil-filled. Two types of idler assembly are shown in Fig. 243.

The motor may be electric or compressed-air operated, and can be mounted on the side or underneath the gear head. With duties up to 70 h.p., a squirrel-cage motor is normally used with direct-on starting equipment. Above 70 h.p., a slip-ring motor is more usual with a stator-rotor starter which greatly reduces the starting current and so the peak load on switchgear and inbye mains. Traction-type fluid couplings may be used between the motor and the gear box where power supply conditions are difficult or it is desirable.

With compressed-air power, the motor is usually of the air-turbine type. The motor inlet-piston valve is operated through a centrifugal governor control on the end of one rotor shaft by a thrust button and rocker arm. When the conveyor is carrying material downhill, the governor nearly closes the inlet valve, so that a partial vacuum acts as a brake and holds the speed within 5 per cent. of normal.

The choice of the width and speed of belt is determined by the following factors: the required peak load delivery per hour, the size of material to be carried and the particular conditions under which the conveyor is to operate. Experience has shown that the peak load per hour may be taken as $1\frac{3}{4}$ times the average hourly tonnage for the actual working period. Assuming run-of-mine coal at the average weight of 50 lb. per cubic foot, the capacity of a troughed belt running at 100 feet per minute may be taken as $W/12{\cdot}5$ where W is the width of belt in inches. The following tables, extracted from N.C.B. Bulletin No. MM. (51)9, are useful guides upon which to

base calculations of drive, horse-power, belt width, weight and number of plies required. The example from the same Bulletin illustrates the method to be used in applying the tables to conveyor calculations.

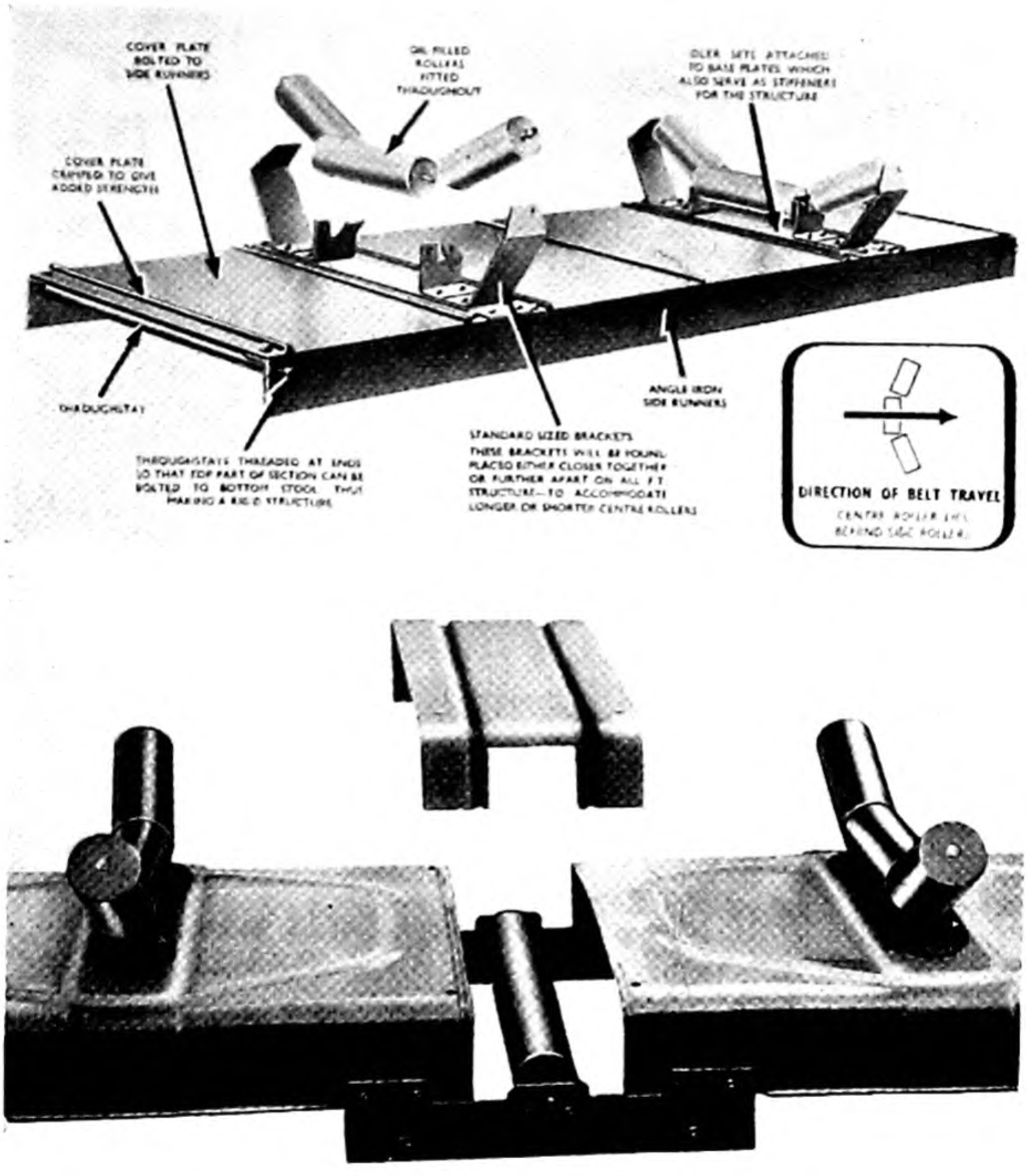


FIG. 243

In gate conveying, the demand for the greatest length of conveyor in one unit, together with the lowest capital cost, calls for belt speeds which may exceed 250 feet per minute. At this speed, with friable coal, special delivery arrangements are necessary to reduce degradation.

Carrying capacity in tons per hour of ordinary troughed belts running at 100 *feet per minute with uniform feed*

Run of Mine Coal		*Belt Width*								
Description	*Density lb. per cu. ft.*	*20 in.*	*24 in.*	*26 in.*	*30 in.*	*36 in.*	*42 in.*	*48 in.*	*54 in.*	*60 in.*
		T.P.H.	*T.P.H.*	*T.P.H.*	*T.P.H.*	*T.P.H.*	*T.P.H.*	*T.P.H.*	*T.P.H.*	*T.P.H.*
Coal free from dirt	50	30	45	50	70	105	145	195	255	325
Medium clean coal	55	33	50	55	80	115	160	215	280	350
Coal containing approximately 20 per cent. free dirt by volume . .	60	35	55	60	85	125	175	235	305	385
Shale	80	47	73	80	113	167	233	313	407	513
Maximum size of normal lumps . . .		6 in.	8 in.	10 in.	12 in.	15 in.	18 in.	22 in.	25 in.	28 in.

NOTES:

1. Capacities for any other belt speed can be obtained by multiplying these figures by $\dfrac{\textit{Speed in ft. per min}}{100}$.

2. The figures in the table are such that spillage would be negligible. However, a belt can be loaded to twice the accompanying figures if it is standing or the practice of 'inching' the belt up is pursued; this, of course, is bad practice, since in all horse-power and belt-strength calculations the belt is assumed to be loaded at the figures shown in the table.

3. Although the quantity of coal carried by a belt is normally a function of the width and speed, it must be remembered that, in this country, there are many seams which produce odd large lumps of coal. The recommended width of belt for run of mine coal containing lumps of certain size is shown in the bottom line of the table. It appears from these figures that, in many cases, we should not use less than 36-inch belting, but this is a matter of economics which can only be decided for each individual installation.

Investigations have been conducted at one colliery in the East Midlands Division, where large coals are common, comparing 30-inch gate conveyors with 36-inch gate conveyors under exactly similar conditions. The additional cost of the 36-inch belt and structure was outweighed by the reduction in labour in cleaning up spillage; furthermore, for a given horse-power and using 32-ounce duck belt, the 36-inch belt could be one ply less and still have approximately the same total strength as the 30-inch belt.

4. For face belts or other flat belts fitted with side-plates, the carrying capacity can be anything from 5 to 10 per cent. greater than the figures shown in the table, depending on the height of the side-plates.

Trunk and gate belt conveyors with fully-troughed structure
Motor horse-power required for empty conveyor running at 100 *feet per minute (for other speeds, multiply by* $\frac{\textit{Speed in feet per minute}}{100}$)

Belt Width	Length of Conveyor—centre to centre							
	500 ft.	1,000 ft.	1,500 ft.	2,000 ft.	2,500 ft.	3,000 ft.	3,500 ft.	4,000 ft.
in.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.
24	1·55	2·7	4·0	5·2	6·3	7·6	8·8	10·0
26	1·7	3·0	4·3	5·6	6·9	8·2	9·5	10·8
30	1·95	3·5	5·0	6·5	8·0	9·5	11·0	12·5
36	2·35	4·1	5·9	7·7	9·5	11·3	13·1	14·9
42	2·72	4·8	6·9	9·0	11·1	13·2	15·3	17·4
48	3·12	5·5	7·9	10·3	12·7	15·1	17·5	19·9
54	3·5	6·2	8·9	11·6	14·3	17·0	19·7	22·4
60	3·9	6·9	9·9	12·9	15·9	18·9	22·0	24·8

Trunk and gate belt conveyors with fully-troughed structure
Motor horse-power required to move material horizontally

Tons per hour	Length of Conveyor—centre to centre							
	500 ft.	1,000 ft.	1,500 ft.	2,000 ft.	2,500 ft.	3,000 ft.	3,500 ft.	4,000 ft.
	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.
50	1·25	2·5	3·75	5·0	6·25	7·5	8·75	10·0
100	2·5	5·0	7·5	10·0	12·5	15·0	17·5	20·0
150	3·75	7·5	11·25	15·0	18·75	22·5	26·25	30·0
200	5·0	10·0	15·0	20·0	25·0	30·0	35·0	40·0
250	6·25	12·5	18·75	25·0	31·25	37·5	43·75	50·0
300	7·5	15·0	22·5	30·0	37·5	45·0	52·5	60·0
400	10·0	20·0	30·0	40·0	50·0	60·0	70·0	80·0
500	12·5	25·0	37·5	50·0	62·5	75·0	87·5	100·0

Trunk and gate belt conveyors with fully-troughed structure
*Motor * horse-power required to lift material vertically*

Tons per hour	Height of Vertical Lift									
	10 ft.	20 ft.	30 ft.	40 ft.	50 ft.	60 ft.	70 ft.	80 ft.	90 ft.	100 ft.
	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.	h.p.
50	0·63	1·25	1·88	2·5	3·13	3·75	4·37	5·0	5·63	6·25
100	1·25	2·5	3·75	5·0	6·25	7·5	8·75	10·0	11·25	12·5
150	1·88	3·75	5·63	7·5	9·38	11·25	13·12	15·0	16·88	18·75
200	2·5	5·0	7·5	10·0	12·5	15·0	17·5	20·0	22·5	25·0
250	3·13	6·25	9·38	12·5	15·6	18·8	21·9	25·0	28·1	31·3
300	3·75	7·5	11·3	15·0	18·8	22·5	26·3	30·0	33·75	37·5
400	5·0	10·0	15·0	20·0	25·0	30·0	35·0	40·0	45·0	50·0
500	6·25	12·5	18·8	25·0	32·3	37·5	43·8	50·0	56·3	62·5

* The table refers only to continuously rated motors. Other motors should not be used without reference to the Electrical Engineer.

*Maximum rated horse-power of motor * to be used in two-drum driving units at 100 feet per minute*

(for horse-power at other speeds, multiply by $\frac{\text{Speed in feet per minute}}{100}$*)*

Width	*32-oz. Duck*					*36-oz. Duck*					*42-oz. Duck*					
in.	*5-ply*	*6-ply*	*7-ply*	*8-ply*	*9-ply*	*5-ply*	*6-ply*	*7-ply*	*8-ply*	*9-ply*	*4-ply*	*5-ply*	*6-ply*	*7-ply*	*8-ply*	*9-ply*
	h.p.	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*	*h.p.*
26	11·5	13·5	—	—	—	13·0	—	—	—	—	14·0	—	—	—	—	—
30	13·0	15·5	18·0	—	—	15·5	18·5	—	—	—	16·0	20·0	—	—	—	—
36	16·0	19·0	22·0	25·0	—	18·5	22·0	26·0	—	—	19·5	24·0	29·0	—	—	—
42	18·5	22·0	26·0	29·5	33·0	22·0	26·0	30·5	35·0	—	22·5	28·0	34·0	39·5	—	—
48	21·0	25·0	29·5	34·0	38·0	25·0	30·0	35·0	40·0	45·0	26·0	32·5	39·0	45·0	51·0	—

NOTES:

1. The above figures are for standard British belting using American cotton duck and are based on the following stresses:
   32-oz. duck—30 lb./in./ply.
   36-oz. duck—35 lb./in./ply.
   42-oz. duck—45 lb./in./ply.
2. The figures are correct for two-drum driveheads and an allowance of 14 per cent. has been made for initial tension and 90 per cent. for drivehead efficiency.
3. For three-drum driveheads, multiply horse-power by 1·06.
   For one-drum driveheads, multiply horse-power by 0·79.
4. A 30-per-cent. increase in the above horse-power may be allowed where all joints are vulcanised efficiently.
5. Horse-power figures are omitted where the belt is considered too stiff to trough correctly.

* The table refers only to continuously rated motors. Other motors should not be used without reference to the Electrical Engineer.

Relation of pulley diameter to belt construction

Smaller pulleys than those given in the following table should whenever possible be avoided in conjunction with the various plies of belt specified.

Number of Plies	*Up to and including 36-oz. Fabric*		*Over 36-oz. Fabric*	
	Driving Pulley min. dia. in.	*Driven Pulley min. dia. in.*	*Driving Pulley min. dia. in.*	*Driven Pulley min. dia. in.*
2	10	8	12	10
3	15	12	18	15
4	20	16	24	20
5	25	20	30	25
6	30	24	36	30
7	35	28	42	35
8	40	32	48	40
9	45	36	54	45
10	50	40	60	50
12	60	48	72	60

The scraper feeder is a short scraper chain conveyor, which receives from the face conveyor and delivers to the gate belt. This feeder replaces the usual return end of the gate belt and incorporates the tail pulley, from which it usually takes its drive. One type is illustrated in Fig. 244, while a telescopic scraper loader is shown in Fig. 245, which also illustrates the stages in the advance of the loader.

(*b*) *Steel link conveyors.* The conveying medium in this conveyor consists of steel plates from $\frac{1}{8}$ to $\frac{3}{16}$ inch thick, turned up at each side, these flanges being fastened between two endless, flat link chains, as shown in Fig. 246. The belt width of these carrying plates is from 20 to 30 inches, the height of the flange being between $2\frac{3}{8}$ and 5 inches and the width of the plates $6\frac{5}{16}$ inches, corresponding to the length of the chain link.

Due to their length being identical, the plates and chain links can turn together round the star-wheels of the driving and return drums. The conveyor structure is usually in 2-yard lengths and consists of a steel angle framework arranged to accommodate top and bottom rollers carrying the top belt and the bottom belt. The rollers are provided with outer flanges to centralise the belt and avoid lateral gliding.

This type of steel link conveyor has been modified in later designs. The bearing rollers are carried on the chains and act as running

rollers, operating on four angle-section travelling rails for the top and bottom link belt as shown in Fig. 247. The running rollers are fitted to the chain—either on or below each twelfth steel plate, and are rotating with the belt. The advantages of this type of construction over the earlier design are:

(1) There is a saving of power of between 40 and 70 per cent. by eliminating the drag of the loaded conveyor on idler rollers.

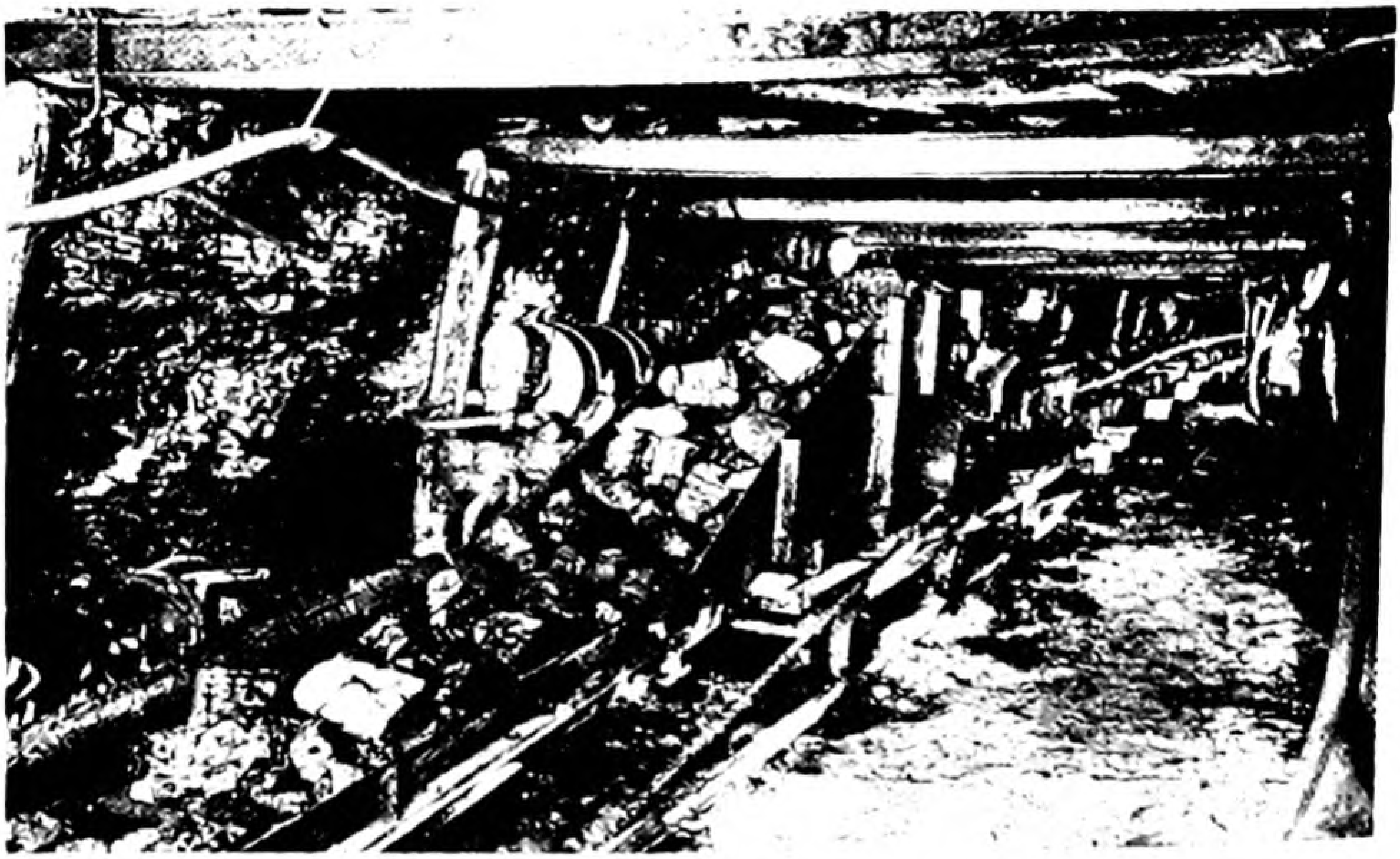


FIG. 244.—10 h.p. scraper feeder delivering onto a gate conveyor.

(2) It is possible to operate a longer conveyor for the same driving power as used in the older type.

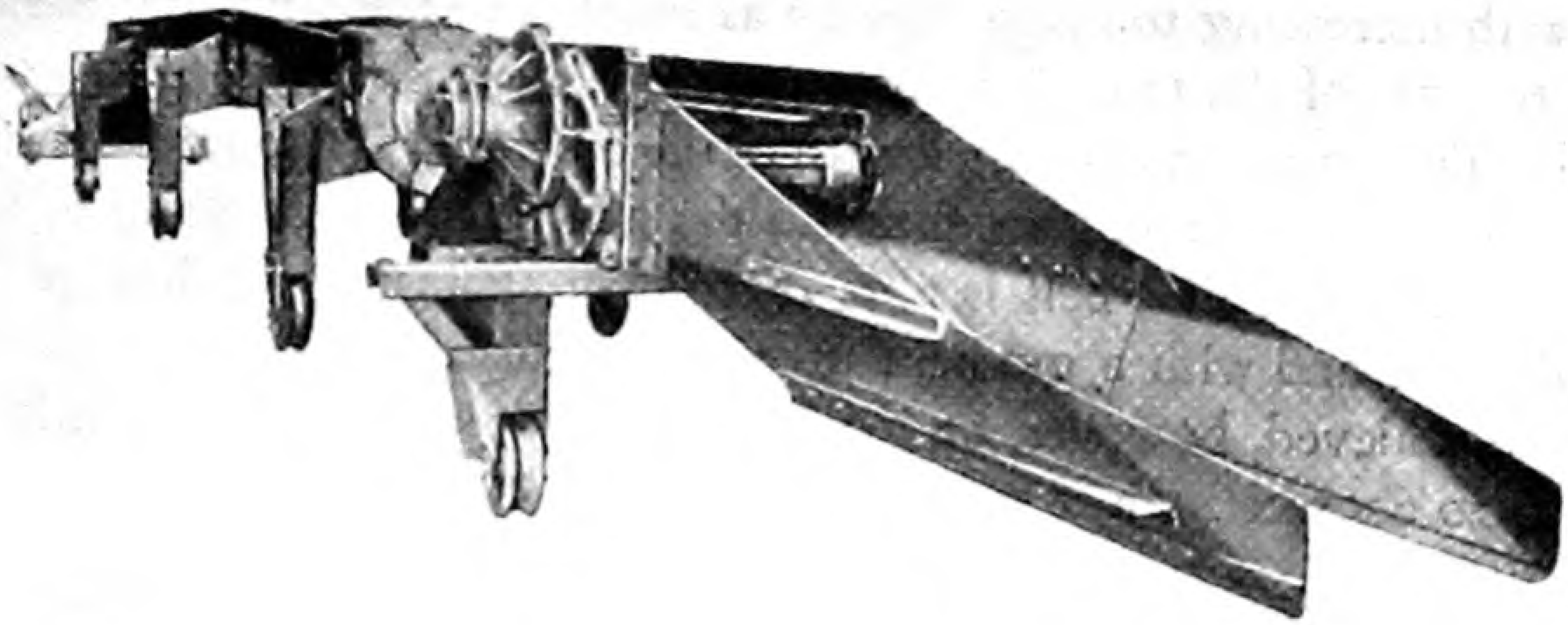
(3) Wear and tear is considerably reduced and the life of a unit is therefore increased.

(4) Due to the reduction in the tension required, the conveyor can operate over more undulating conditions.

(5) The conveyor has greater flexibility and can be run with small deviation in alignment.

(6) The coal is handled more gently, since the repeated bumping over the idler rollers is eliminated.

The chains must take the tractive force of the drive and, between the brackets, the weight of the plates and rollers as well as the material carried. The chains are therefore of very heavy construction, having a tensile strength of 48,500 lb., which in certain cases



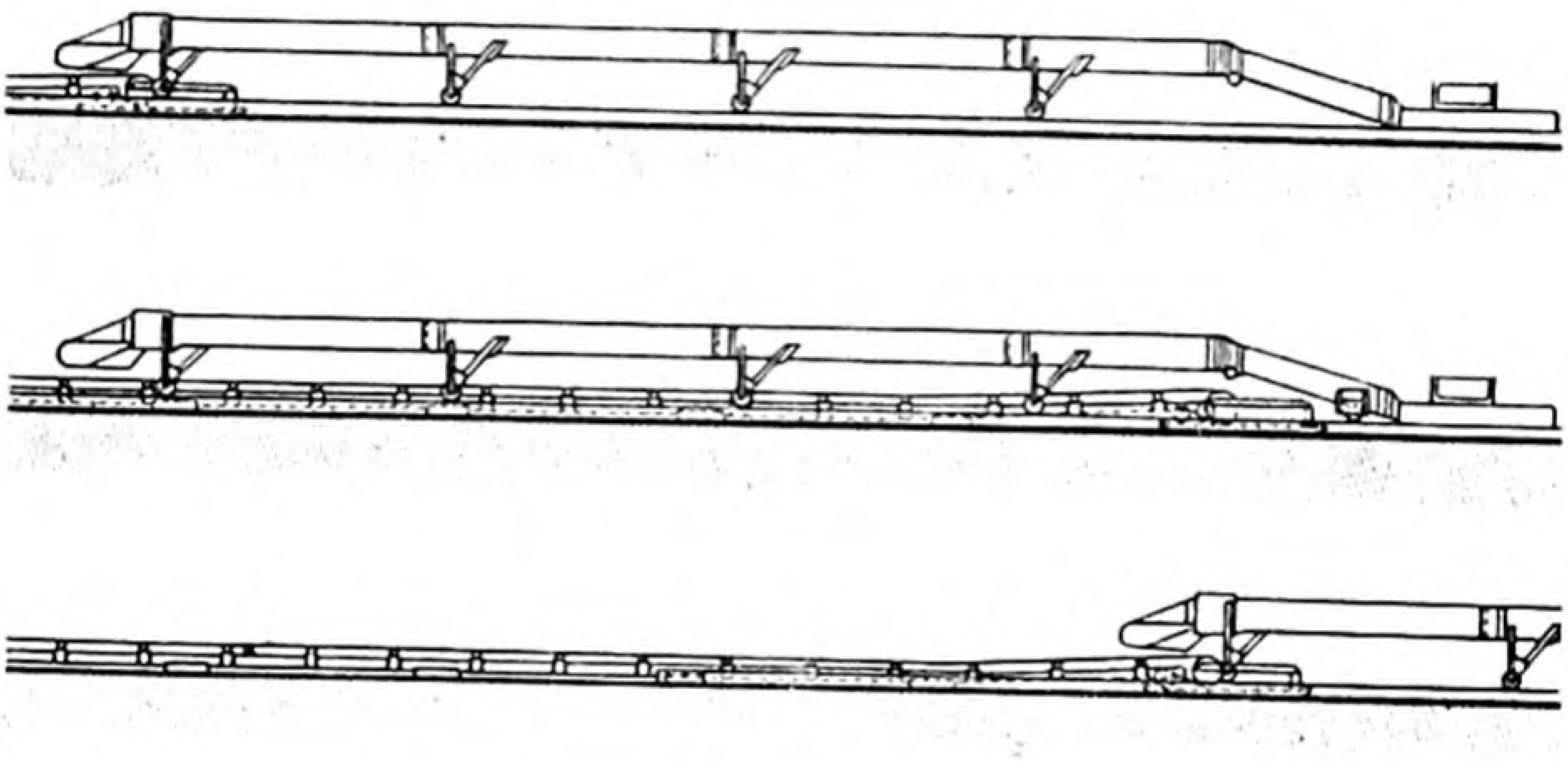


FIG. 245

FIG. 246

with increasing tonnage may be as much as 77,000 lb., the tensile strength of the two-chain belt therefore reaches 97,000 or 154,000 lb. The normal maximum length of the conveyor is about 440 yards. If angle-bar sections are fitted to the belt to prevent rolling of the material, the link belt is able to convey against a gradient of 40 degrees and with a gradient of up to 25 degrees. The best results are achieved, however, on gradients from 15 degrees with the load to 20 degrees against the load.

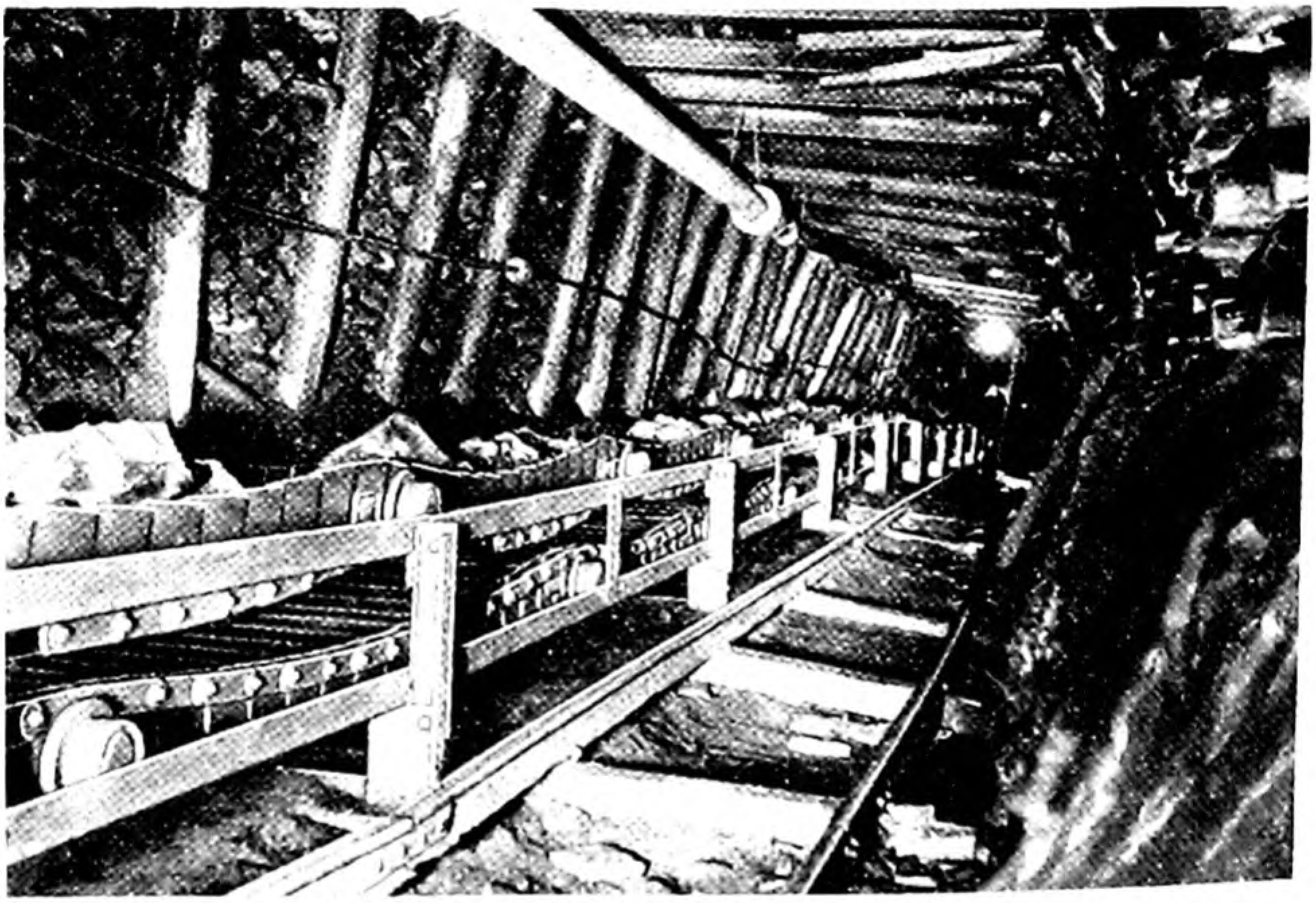


FIG. 247

Due to high inertia of the large masses to be moved when starting up the conveyor, the steel link belt requires a large driving motor. The squirrel-cage type is normally used since it can be started direct 'on line'. With electric drive, a flexible coupling is required to obviate damage to the chains when starting; with compressed-air motors, regulation of the starting speed can be carried out at the inlet valve. The driving gear is either roller chain or 'V' belt, the latter having the advantage of absorbing starting shocks. In the modern link belt conveyor the drive is often direct to the driving drum through bevel or spur gearing, as shown in Fig. 248.

The return drum also serves as a tension pulley. The tension

device consists of a steel angle frame and two guide bolts. A stretcher bar between the tension bolts is coupled to the shaft and bearings of the return drum, which can be adjusted for tension with the male screws on the stretcher bar. The return drum can be roughly tensioned, using chains and turnbuckles, the return drum assembly being anchored to stay bars. The height of the driving head may be up to about 4 feet 8 inches in the older type, but is considerably reduced in the new type design. As is usual with large

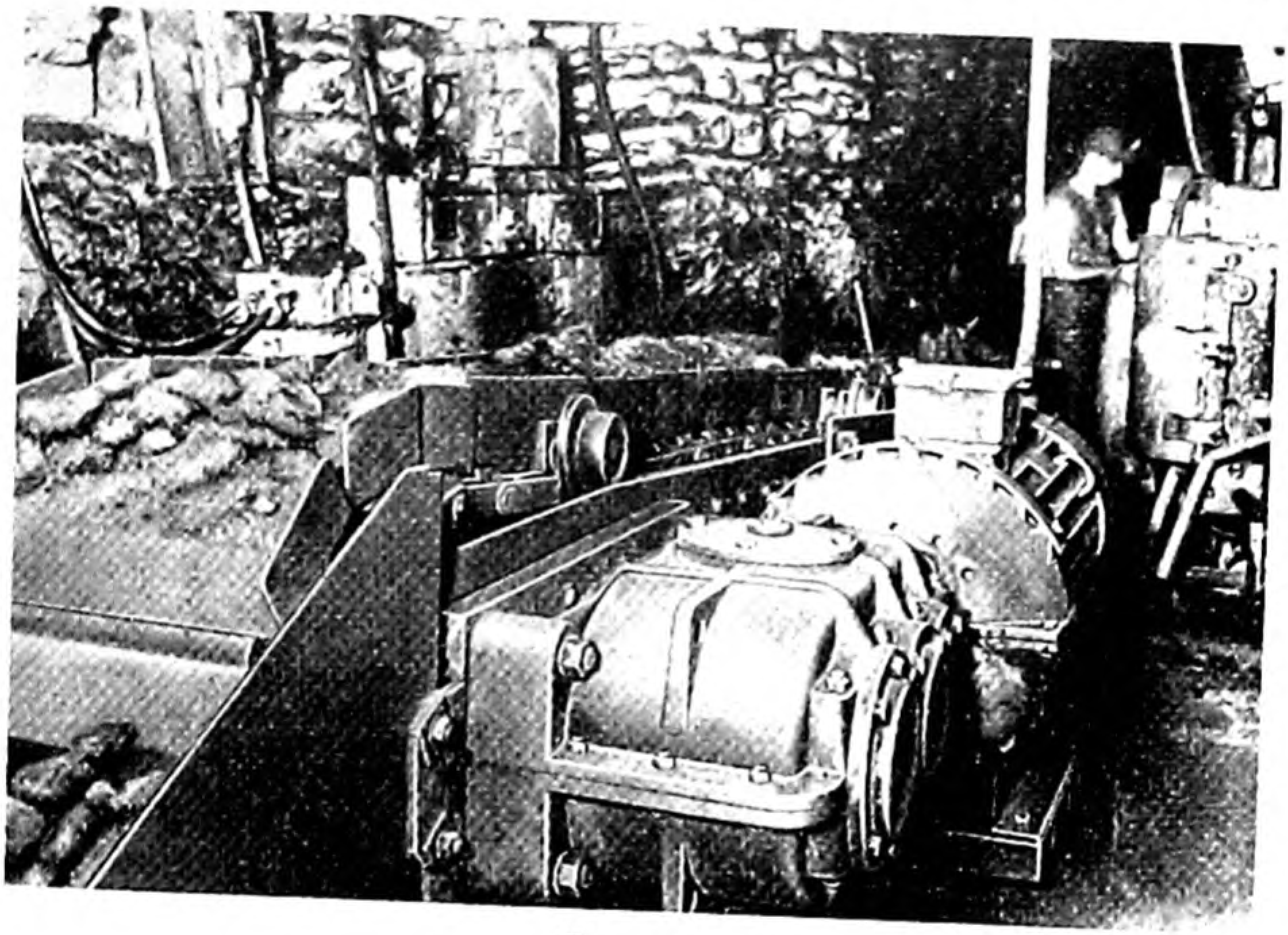


FIG. 248

drives, the motor and gear are set at the side of the driving head. Where the power required is up to 30 h.p., however, an electric drive can be used which is housed inside the driving drum, reducing the overall height to about 3 feet.

The wear of the steel link belt depends upon its running speed, the materials to be transported, and on the maintenance efficiency, especially with regard to the chain. The wear is considerably greater in link belts of the older design. A conveying speed from 120 to 150 feet per minute should not be exceeded, the maximum speed is about 180 feet per minute, but to run at this speed increased wear results. The lower speed of this type compared with the rubber

belt conveyor is compensated by its larger capacity, the conveyor being capable of dealing with outputs up to 250 tons per hour.

Where waste material is conveyed, an increase in the wear must be expected; with fine-grained, wet stone, the dust gets into the link chain connections and the roller bearings. With coal haulage, the life is about four years, and with waste conveying, two years.

The operational cost is rather less than for rubber belt conveyors of equivalent length. The link belt has the advantage that heavy materials and machines can be transported fairly easily and without damage to the conveyor or structure. This conveyor has also a more extended use in inclined conveying both to the dip and to the rise than rubber belt conveyors, while the lengthening of the conveyor is relatively easier. The minimum height requirement of about 3 feet for the drive head is a disadvantage.

The curve belt conveyor. A recent introduction in gate conveyors is the steel link belt designed to operate round curves. The conveyor, which is manufactured by a German firm, Hemscheidt (Wuppertal), can negotiate a curve of radius as small as 18 feet. The belt consists of individual troughed steel sheets, attached to each of which are three bearing rollers and a central guide roller each $2\frac{1}{2}$ inches in diameter. The bearing rollers offer a three-point support to the troughed steel plates. The troughed steel sheets, which have a trough width of 2 feet $4\frac{1}{2}$ inches, are connected to each other on both sides of the conveyor by a link chain of 0·63-inch diameter section. The adaptability of the conveyor to negotiate curves is provided by a bifurcated sheet in the centre of each troughed plate supporting the guide roller, and on the other end a gable fitting alongside a round bolt fixed to the next trough. The bearing rollers run inside two channels with vertical web, the guide roller in a channel with horizontal web, as shown in structural part of Fig. 249.

Rubber strips 0·28 inches thick, 4·9 inches wide and as long as the trough is wide, are inserted into special grooves below the steel plates to close the slits between the individual steel plates when the conveyor is negotiating a curve or spiral, as shown in Fig. 250.

If the haulage capacity is 200 tons per hour, the power required for a level, straight roadway will be from 15 to 20 h.p. per 200 yards. The driving station may be located at any point on the conveyor, and the number of driving-points is unlimited so that any length of conveyor can be installed. This feature is made possible by the use

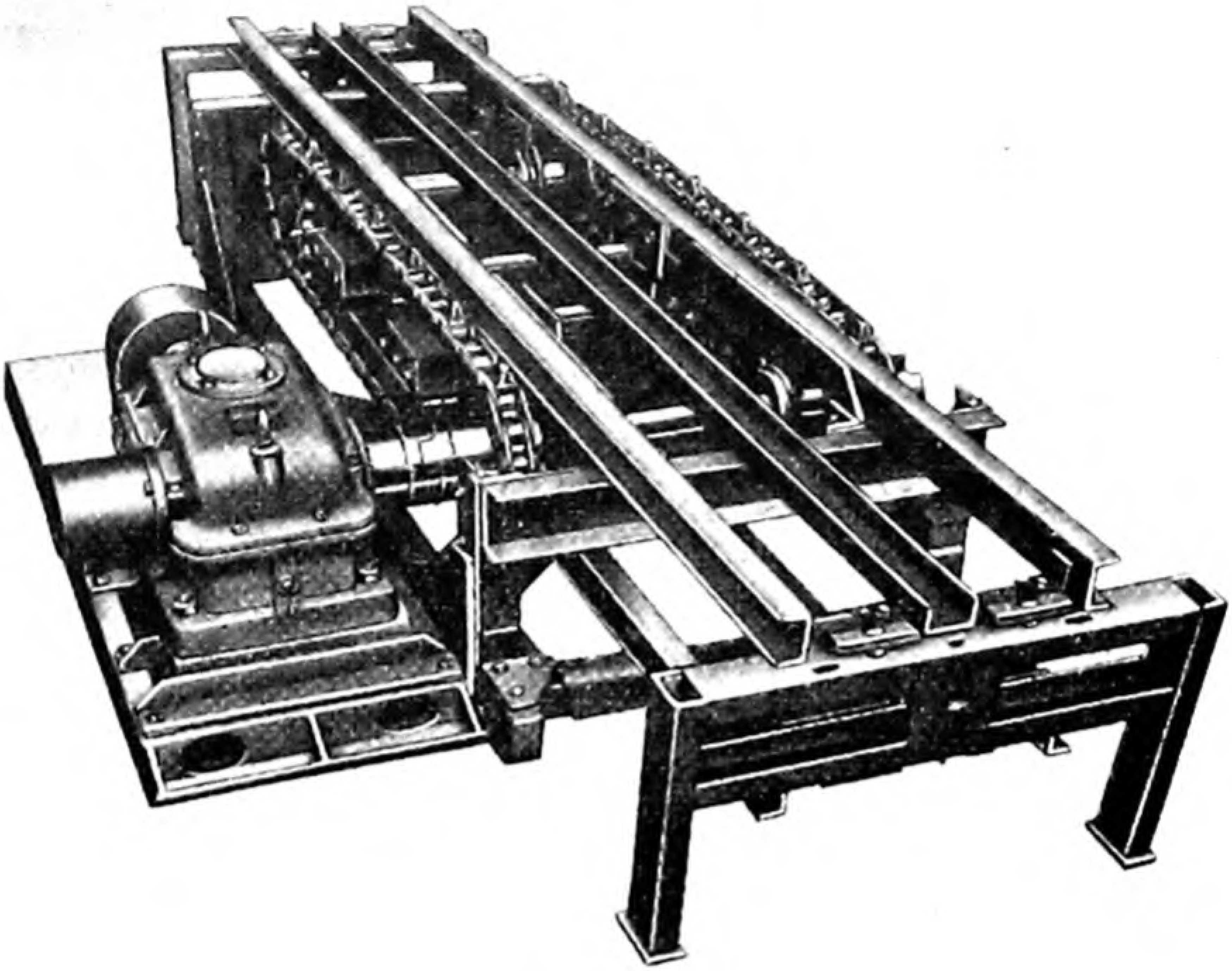


Fig. 249

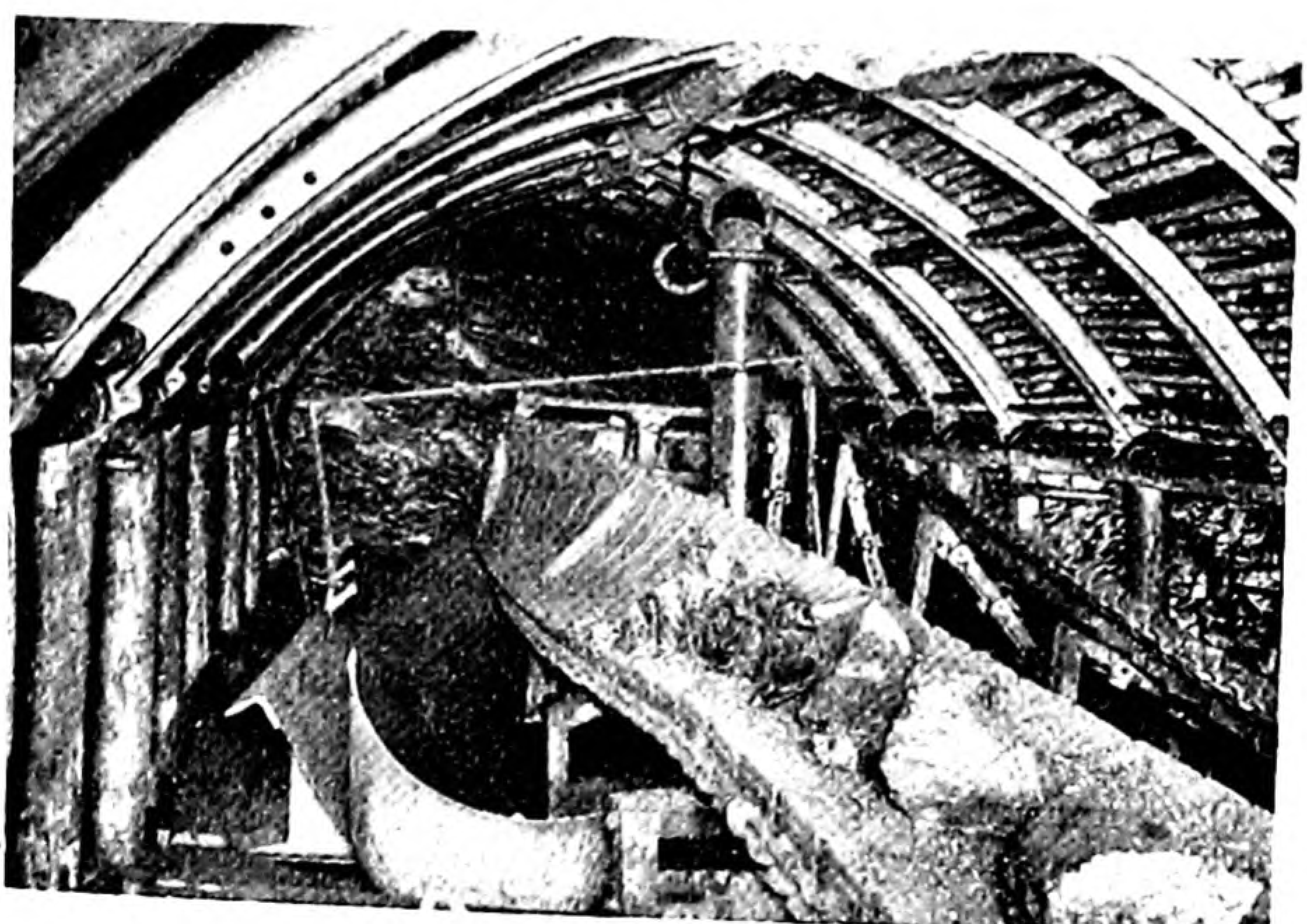


Fig. 250

of a caterpillar-chain drive, which is geared to the link-chain of the conveyor, as shown in Fig. 249. The conveyor drive may operate on the normal top belt or the bottom belt, or both may be driven at different driving stations. The curve belt can be turned on its longitudinal axis so that convenient discharge-stations can be created at any point, and both sides of the conveyor can be used for haulage purposes.

In addition to gate-road haulage, where the special advantages of the curve belt conveyor can eliminate transfer stations because of its unlimited length (using drives 200 yards apart) and both runs of the conveyor can be used for the simultaneous conveying of coal outbye and waste or material inbye, the conveyor has also been used satisfactorily on longwall faces. This introduces quite new possibilities for the extended use of this form of conveyor, the effect of which cannot yet be predicted. The conveyor may be used as a combined face and gate-road conveyor and the transfer station at the face eliminated, while the return belt can be used for materials or waste haulage. The conveyor would negotiate the face and gates in a complete circle system. The curve of the conveyor from a longwall face is shown in Fig. 251.

Within the face length of the run of the conveyor, the framework is extremely low because the return belt does not need to be placed underneath. Sections of the face framework may be armoured if necessary, for use with a mechanical coal-winning machine. A later modification is to introduce spring-cushioning between the face sections to allow deviations in alignment of up to 3 degrees horizontally and 5 degrees vertically.

In the case of a face working with a prop-free front, the curve belt will either be advanced as a whole or in stages, as will be the case where a coal-planing or plough machine is in use. In this case, a telescopic section of conveyor, allowing a 2-yard advance, is incorporated in the framework at the face-end of the gate road. The belt can adapt itself to this increase in length, since the link chains connecting the troughed sections invariably sag during normal operation.

The output of a single steep face is often too small to make a complete conveying system pay, and then it would be necessary to connect two or more faces to a mother belt. Of course, under these circumstances delivery points cannot be avoided and this makes the

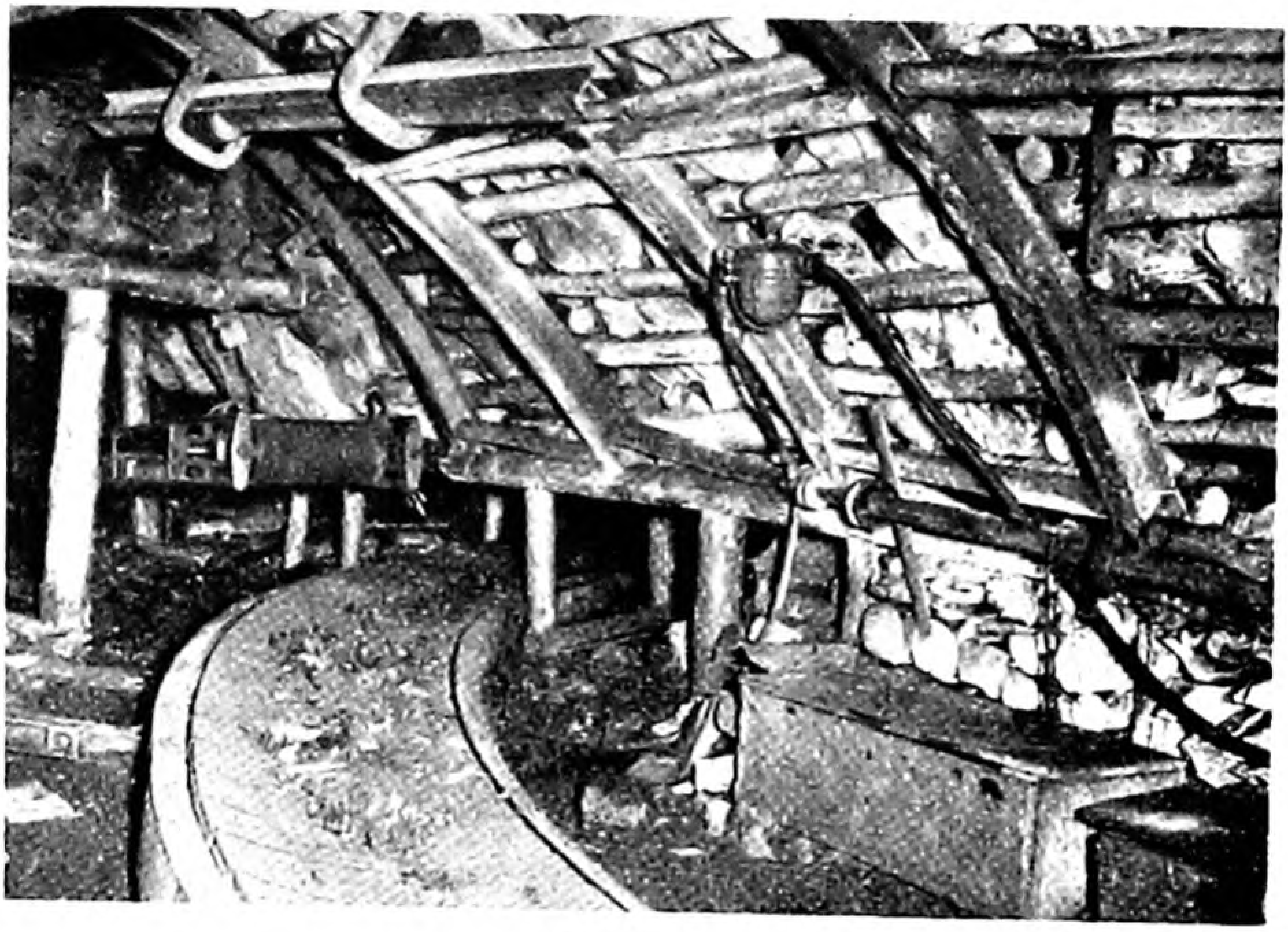


FIG. 251

conveying of waste and materials towards the face by the same conveyors a much more difficult problem.

All these difficulties can be overcome by the recently introduced belt units of about 100–120 yards in length, constructed on the lines of the curved-belt conveyors, driven by motor units at distances shorter than the length of the belt unit and running on the frame structure extending over the full length of the roadway. By means of switches, it is possible to use one belt unit for more than one face, or to use more belt units on the same frame structure.

The construction of the new Hemscheidt belt-unit conveyor is simplified by replacing the two connecting belts by one special link chain in the centre of the sheets.

Other makes of curve-belt conveyors are the Aumond, Eickhoff and Hauhinco conveyors, which can manage curves of 75-yard radius.

Section 2. Rope Haulage

(*a*) *General introduction.* Rope haulage of mine cars for the transportation of coal or debris is not common in the flat formations. This form of haulage is not inferior to the belt conveyor with regard

to technical efficiency, but is more expensive. The belt conveyor has the advantage of continuity of haulage and facilitates the use of large mine cars, reducing the rolling stock required and increasing the number of possible mine-car trips. In steeper measures, where the inclination is over 40 per cent., rope haulage on the dip is widely used. In the flatter measures, rope haulage has often stood the test as an additional or auxiliary means of haulage for materials and machines for which the rubber belt or steel belt are not suited.

(*b*) *Rope winches*. The haulage engines for winches or hoists are

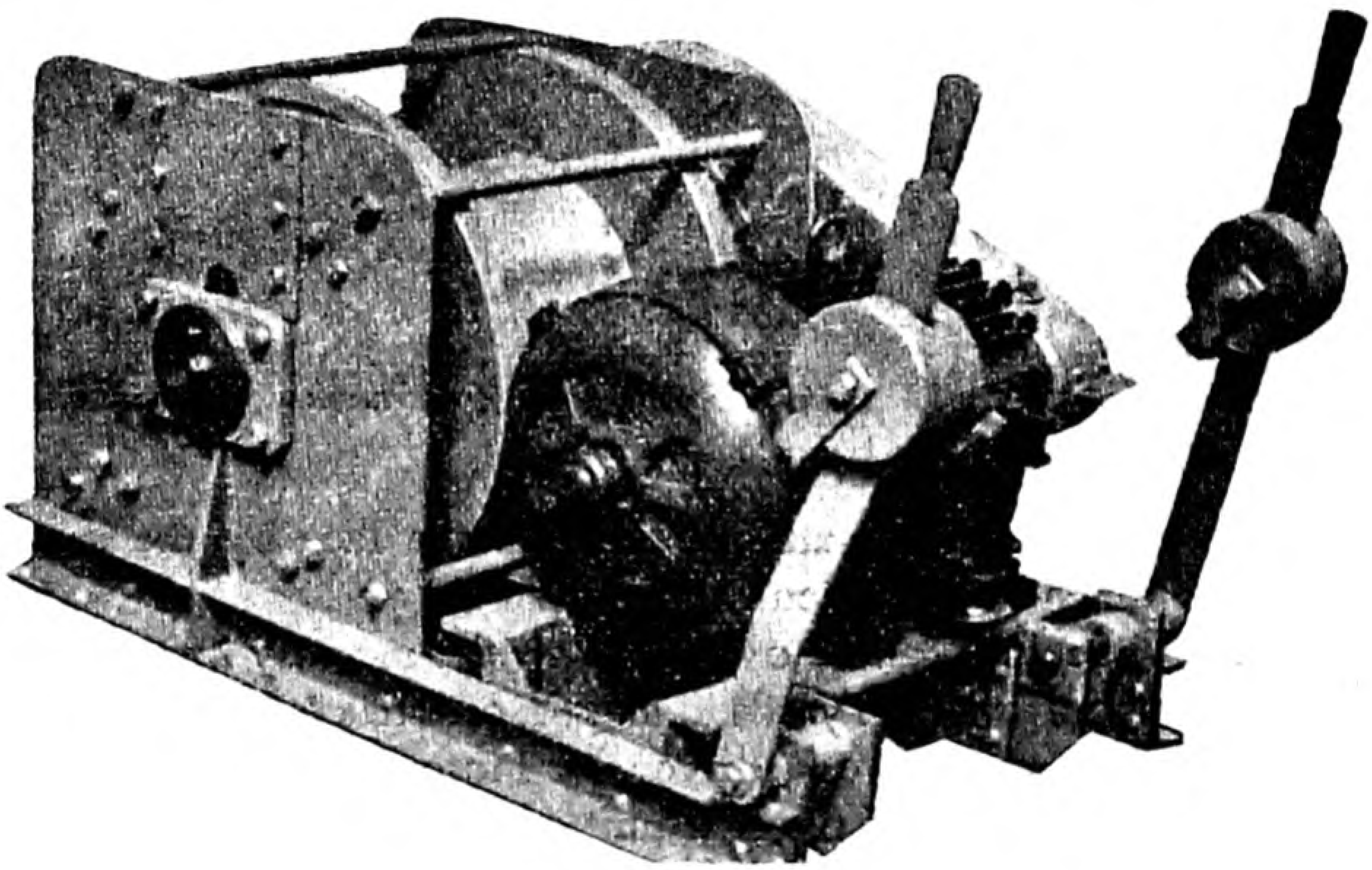


FIG. 252

usually of from 8 to 15 h.p., driven either by compressed-air turbines or electricity. In the case of smaller hoists, they are placed inside the drum itself, while in larger hoists they are set at the side of the drum and driven through a double transmission gear. An electrically operated winch is illustrated in Fig. 252.

The winches can be set at floor level, but it is preferable to install them on steel girders set below the roof and about six feet above the floor. This arrangement has the advantage of reducing interference to the traffic along the road and avoiding damage to the winch in the event of overdrawing the trains from either side of the winch, without the installation of guide rollers which are required for side location at floor level.

There are two distinct layouts for the location of the winch haulage system. The commonest layout introduces two winches, one at each end of the road along which trains are to be hauled. The hauling winch operates the main rope, pulling the train, while the second winch runs out the tail rope fastened to the last car on the train. On the return journey the position is reversed. The greatest distance within which two winches located in this way can operate is about 600 yards. It is possible, by shifting the rope from the first to the last car, to draw the train past the winch position and thus

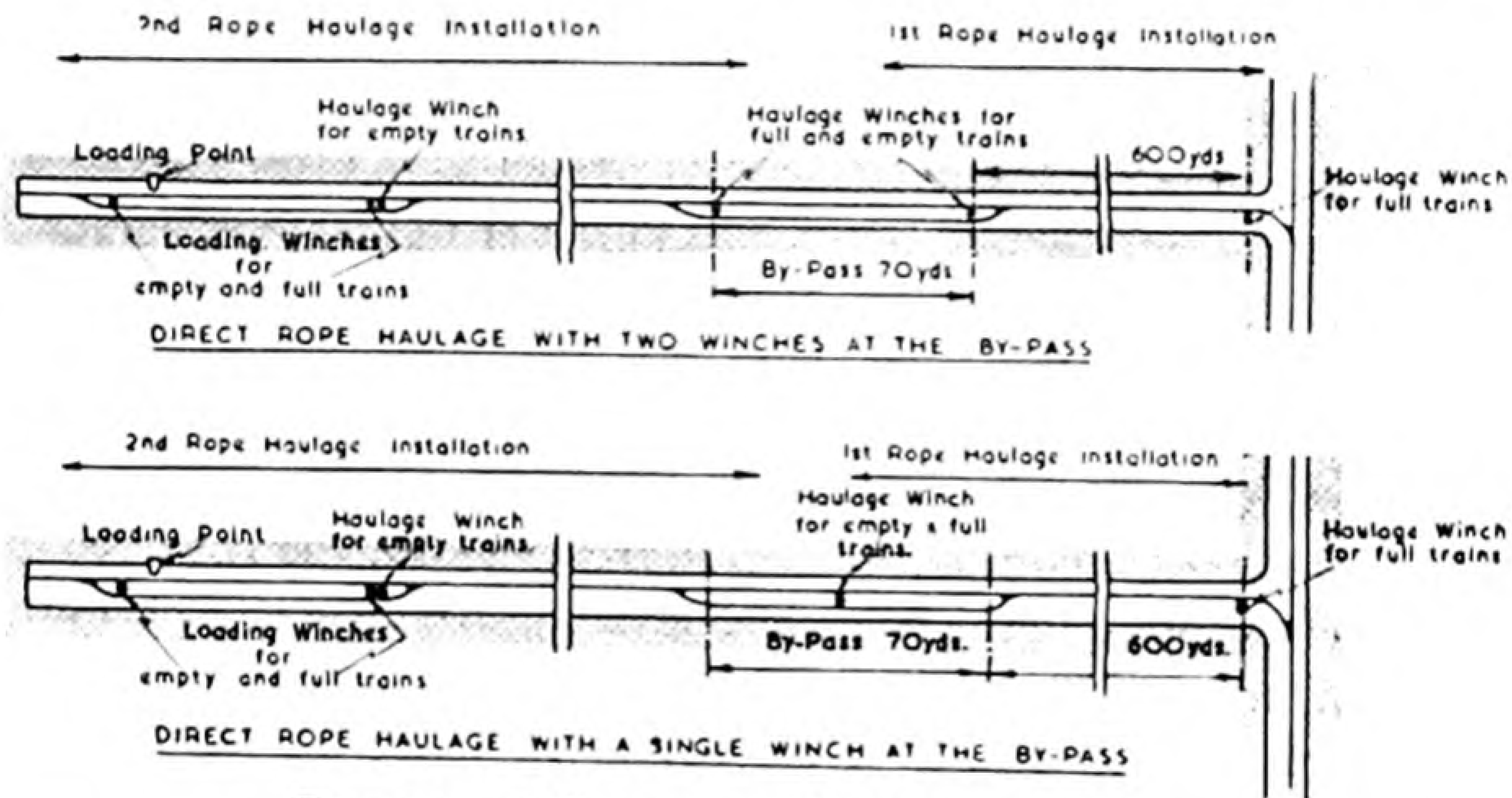


FIG. 253.—Different types of direct rope haulage.

increase the length of haul to about 660 yards. This length includes the distance of 600 yards and the length of the train, which is taken to be 60 yards. Single track is sufficient over the greater part of the road, double track being provided near the winch to take one full and one empty train side by side.

Where the gate road is longer than 660 yards, a second pair of winches can be used in the following section as shown in Fig. 253. In this case it is sufficient to place the two winches in the middle of the roadway and within 50 yards of each other, since this distance can be covered by changing over the main ropes to the back of the trains.

The capacity of this form of direct-rope haulage is dependent on the speed of the haulage, the length of the road, the number of cars per train and the size of the cars. The time spent on rope-changing is also an important factor. Generally, an average speed of about 3 to 4 m.p.h. can be maintained with 30 one-ton capacity cars per

train. Under normal conditions an output of 1,000 tons per shift can be hauled over 300 yards and 600 tons per shift over 700 yards with winches of 15 h.p. With increasing length of haul and using the repetitive system described, i.e. the two pairs of winches, the haulage capacity is about 600 tons per shift, which remains almost constant and independent of the distance hauled. With only one additional winch on the longer haulage distance, the system has not the same continuity and the capacity is reduced by about half.

Additional small winches may facilitate the gathering of trains at the ends of roads, i.e. at the loading point as well as near the junction of the gate road with the main road, so that the cars can be more quickly brought to the main winch haulage.

One operator or driver is required per winch and one haulage man per train, while two men are required at the loading station, assembling empty cars and full cars, and one man coupling and uncoupling the trains.

It is possible to adopt the normal main and tail-rope system of operation. With this system a greater length of rope is required and a double-drum winch is used, the return pulley for the rope being at the farthest point of the haul. The advantage of this method is the saving of one winch operator and the ease in increasing the length of haulage as the gate road develops, since the winch is static. The system has the disadvantage that the maintenance is increased and the transference of trains to a second pair of winches is difficult or impossible at the return-wheel end of the haul. This system is ideally suitable where rope haulage is to be installed for the transport of materials and supplies.

Section 3. Locomotive Haulage

(*a*) *General introduction.* Locomotive haulage in gate roads will be considered only when conveyor transport cannot be used for either economic or technical reasons. When the output to be handled is small, the transportation cost per ton would be excessive if conveyor transport were used, while its introduction would not be practicable in gate roads which are not reasonably straight. These difficulties rarely occur when flat seams are being worked, but in the steeper formations these conditions often arise. In the latter case, the output per face unit is usually between 100 and 200 tons per day and the gate roads must follow the changing inclina-

tion of the seam, in which case rope or locomotive haulage is the only means of mechanised transportation. Locomotive haulage has the advantage of greater flexibility and can operate from several gate roads utilising a single locomotive, which can also be used for shunting at main loading stations in subsidiary or main roadways.

The essential requirements for a locomotive for gate-road transport, other than the necessary precautions against the presence of methane, are that it should be small and easily dismantled to allow it to be carried through staple shafts when necessary. Because of these requirements, only compressed-air, battery and diesel locomotives can be considered. The trolley-wire locomotive is excluded because of the increased possibility of the presence of methane near the coal face and the difficulty in hanging trolley-wires at the required minimum height in gate roads suffering from excessive strata pressure.

The small gate-road gathering locomotives, whether they are of the compressed-air, battery or diesel type, are usually from 8 to 15 h.p. and are capable of operating trains of from 20 to 30 mine cars of one-ton capacity at a speed of about 4 m.p.h.

(*b*) *Compressed-air gathering locomotive.* Like the main-road locomotives, these also consist of a carriage or frame, air containers, 'working bottle', motor and gearing, and the cab, which can often be removed or tilted over. The number of air containers is usually four or five, the air pressure being kept to at least 200 atmospheres, at which pressure less difficulty is experienced with the containers than is the case with the larger main-road locomotives. The motors are either single-cylinder engines, one being placed on each side of the locomotive, or the high-speed four-cylinder type, in which the pistons are arranged in pairs on a double crankshaft. Motors having five radially placed cylinders round the crankshaft are also being introduced. In the case of the high-speed motors, the drive is transmitted to the axles through a flexible coupling, roller chain or gearing. A Demag compressed-air locomotive is shown in Fig. 254. This locomotive is 4 feet 2 inches high, 2 feet 10 inches wide and 6 feet long.

(*c*) *Diesel gathering locomotive.* The illustration in Fig. 255A shows a German 10-h.p. diesel locomotive, equipped with a horizontal four-cycle diesel engine without compressor, and in Fig. 255B the British 24-b.h.p., $2\frac{3}{4}$-ton, Hunslet 'pit-pony' locomotive. The Huns-

let locomotive is a four-wheeler, with 16-inch wheels spread over a base of $24\frac{1}{2}$ inches, and is provided with a single-speed gearbox. This locomotive can negotiate curves of 15-feet radius and has a speed of $6\frac{1}{4}$ m.p.h. The safety devices incorporated at the suction and

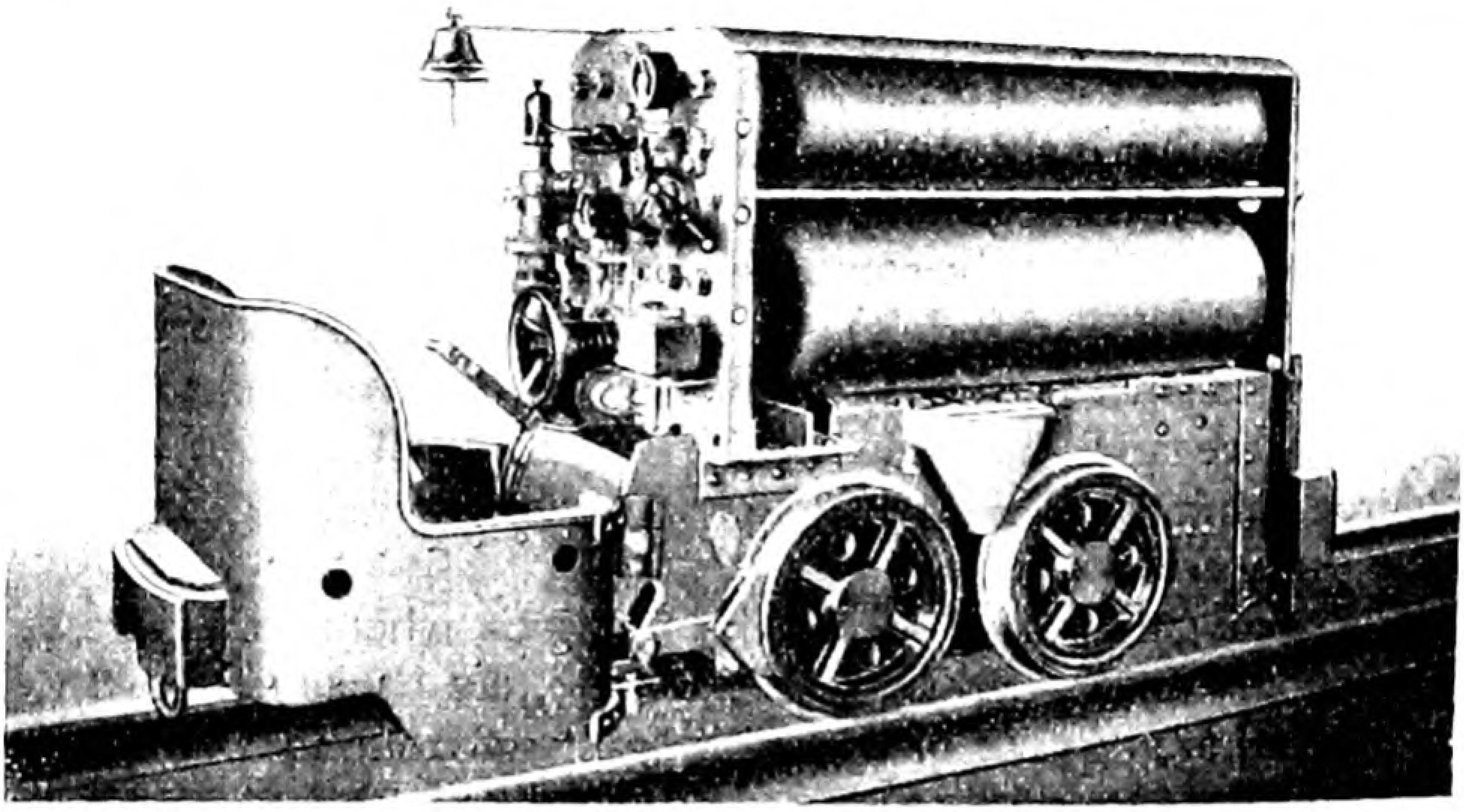


FIG. 254.

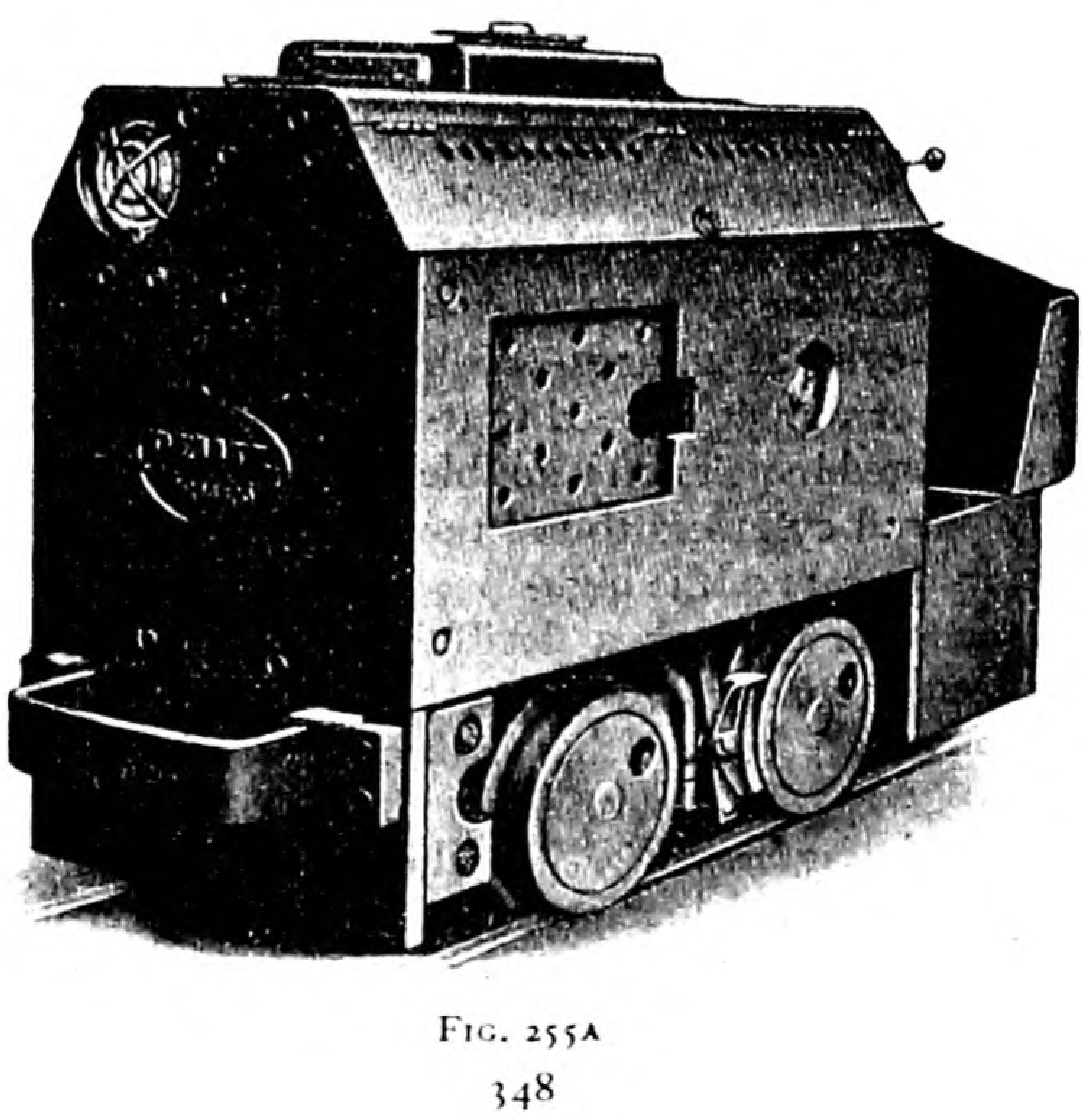


FIG. 255A

exhaust sides and in the fuel container are identical with those described for the main-road locomotive described in Part III, Section 1. The fuel tank generally has a capacity of from 4 to 5 gallons, which is normally sufficient for a double shift.

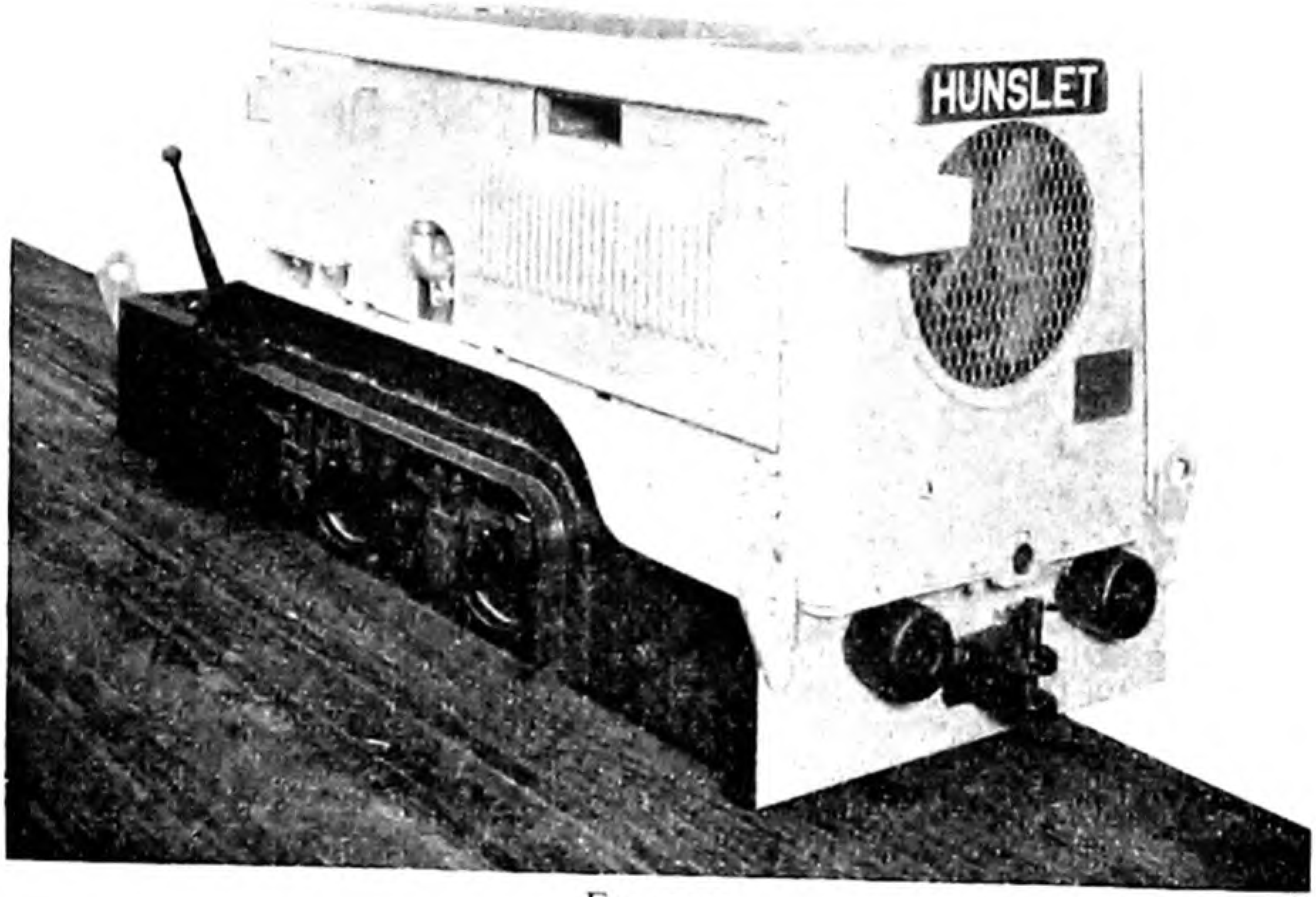


FIG. 255B

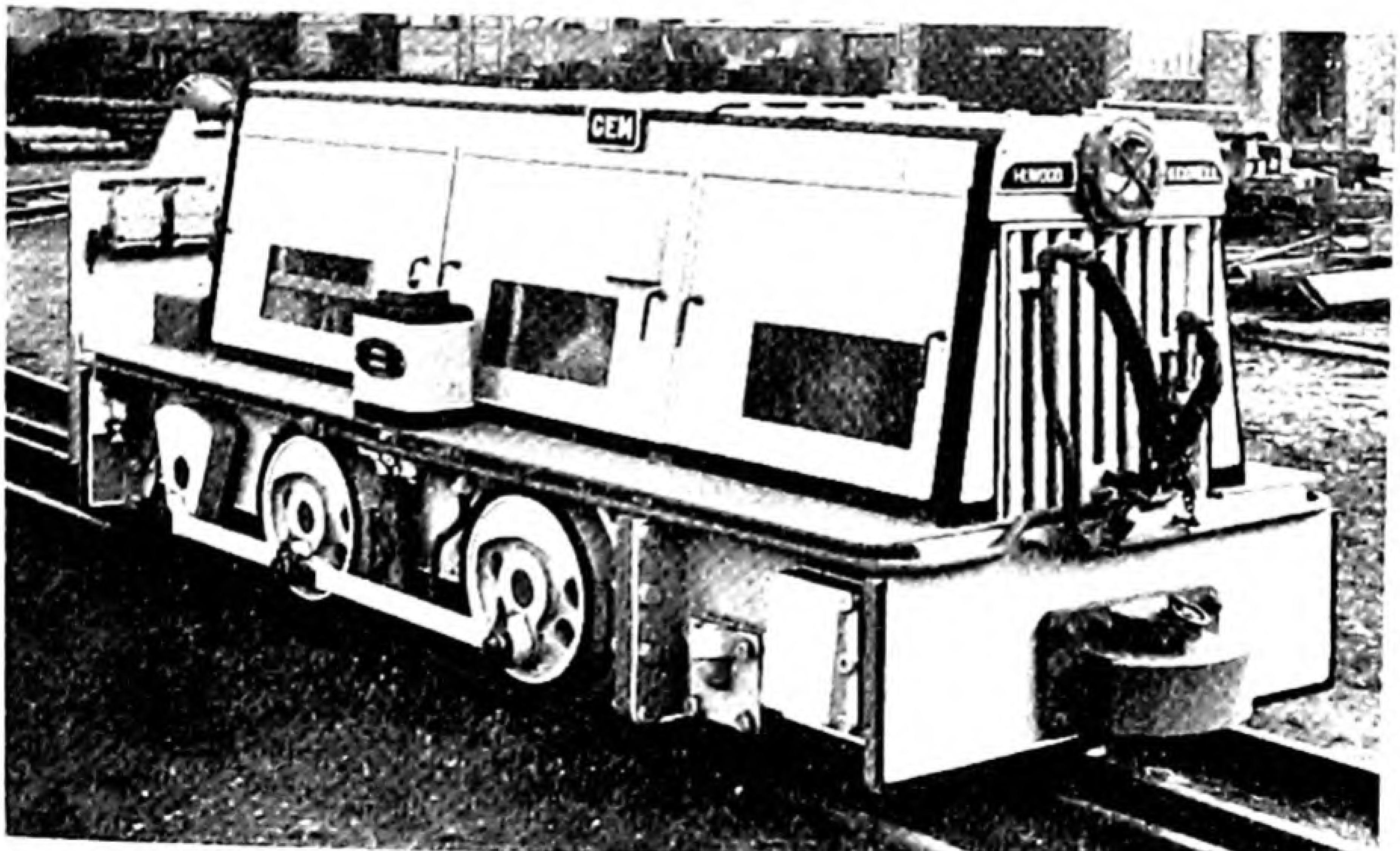


FIG. 255C

Another British locomotive of the same class, the Huwood Hudswell 25/34 b.h.p. locomotive, is shown in Fig. 255c.

(*d*) *Battery-operated gathering locomotive.* The battery gathering locomotive differs from the main-road locomotive mainly in the smaller dimensions, as shown in Fig. 256. This smaller type has one motor only instead of two, while the battery and traction switch are

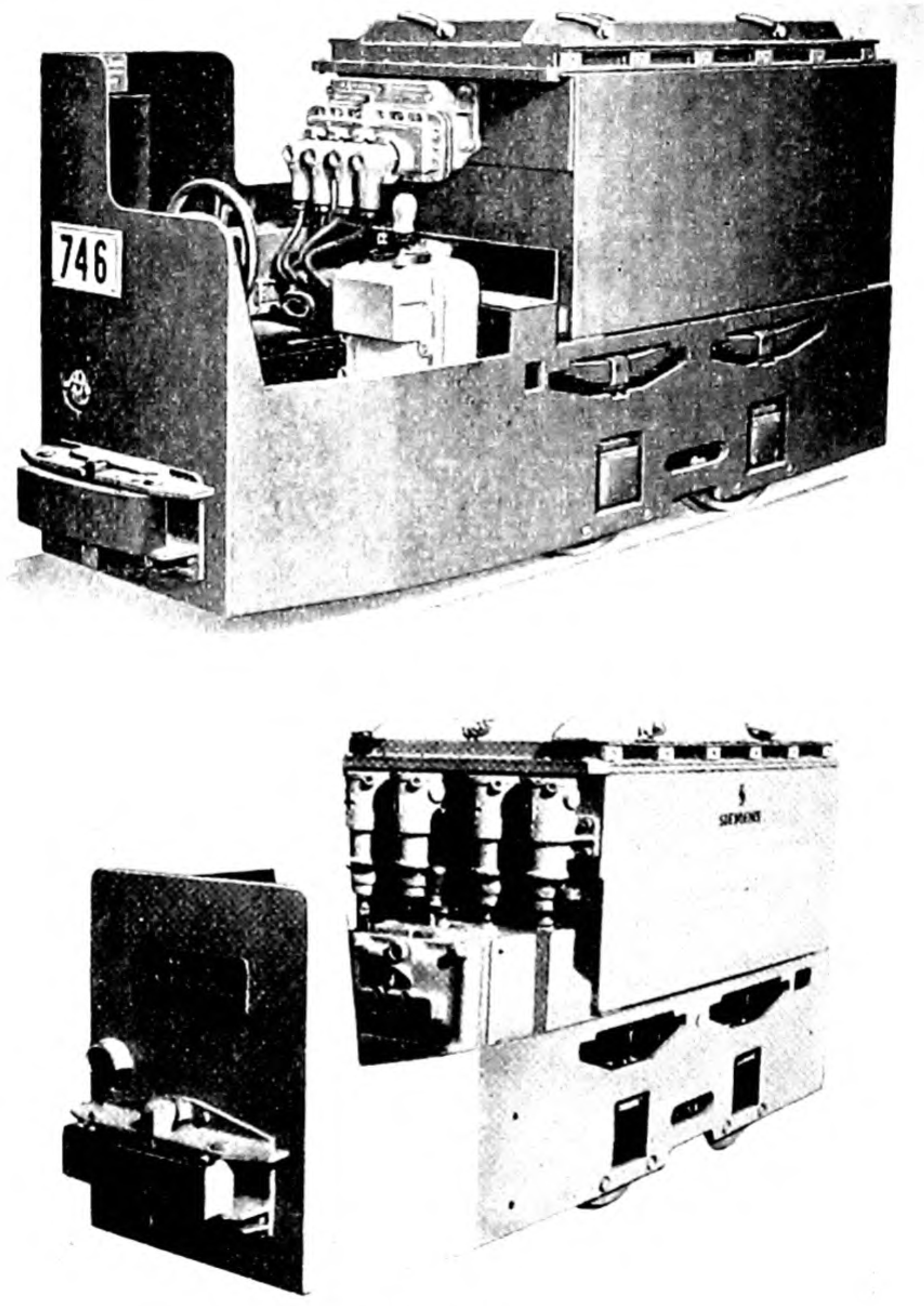


FIG. 256

both equipped with flame-proof protection. In order to obtain a high efficiency with the least possible weight, grid or armoured battery plates are preferred. The capacity of the battery has to be rated in such a manner as to secure a sufficient reserve of power for one shift. The battery replacement is usually carried out effectively by transporting the batteries in special battery cars to the pit-bottom changing station and carrying out the replacement on changing tables in the district cross-cuts. If the distance between the gate roads and the shaft is small, it may be possible to bring the locomotives to the battery-charging station at the pit bottom. It is also possible to have the battery in two equal sections and to keep the locomotive in operation over two shifts without replacement or battery charging. The working capacity of an 8-k.w. locomotive is about 250 gross ton miles.

(*e*) *The choice of locomotive type.* In order to make the best choice between the three types available, the ventilation factor, initial cost and operational cost must be considered. The compressed-air locomotive offers absolute safety with regard to the presence of methane, the others usually offering adequate safety in normal conditions. The initial capital cost of the compressed-air locomotive is the greatest and the diesel locomotive the least. The additional installation cost required for a compressed-air locomotive adds considerably to the initial cost; this applies to a lesser extent in the case of battery locomotive haulage, for which a battery-charging station is required. Additional equipment is not required in a diesel locomotive installation. If, however, these installations already exist for main-road haulage, the choice may be in favour of standardisation throughout for gathering and main-road locomotives. The operational cost is highest for the compressed-air locomotive, while it is lower and about the same for each of the other two types. The operational cost for locomotive gate-road transport can be assumed to be about ten times higher than for similar main-road transport with the same type of locomotive, this difference being based upon the shorter length of haul and the smaller pay load per locomotive, i.e. the lower number of ton-miles per locomotive-shift.

Variations will occur in the initial capital cost and operational cost, depending upon the number of locomotives in use. In the case of a small installation, diesel locomotives will be given the preference, whereas battery or compressed-air locomotives may be con-

sidered for a larger installation. Other factors may also be important, such as the contamination of the air by the diesel locomotive, the smell of the exhaust gas and the heating effect on the air current. The compressed-air locomotive has the advantage of cool operation and provision of additional dry fresh air from the exhaust.

PART II

TRANSPORT IN STAPLE SHAFTS

Section 1. *General Introduction*

The work involved in staple-shaft transport is difficult and can, in large mines, be an even greater task than main-shaft transport in smaller mines. The duties of the staple-shaft transport system have to include coal, waste and materials transport, as well as man-riding facilities.

Coal-hoisting to an upper level is only applied in dip-working or in other special cases but, normally, waste and supplies are taken down to the lower level. The distribution of waste material and supplies takes place at the main winding level and therefore must be taken up through the staple shafts to the coal faces lying between the winding and the ventilation levels. Where the ventilation level is also a supply road, the waste material and supplies from the surface are sent down the staple shaft. The shaft also has to be equipped for man-riding.

The tonnage of coal to be handled per staple shaft per day varies within the wide limits of 100 and 800 tons and there are cases with outputs as high as from 1,000 to 1,500 tons per day. Similar, though generally smaller, quantities of waste also have to be handled. The quantity of materials and supplies to be dealt with in the staple shafts depends on whether the supports in the district are of timber or steel. With timber the supplies are large, while with steel, which can be re-used, the quantity is much less.

Depending upon the local conditions, there are staple shafts serving for both coal and waste transport, either in the same or different shifts. This is generally the case in steep-seam working. On the other hand, the staple shafts may serve only for the transport of either coal or waste and materials, especially in flat seams, but these separate duties will only last a certain time, when eventually a staple shaft will become used for either the transport of waste

entirely or converted to coal transport. Because of the probable change in duty, this factor must be considered when staple shafts are originally driven and equipped.

Hoisting in staple shafts can be by cage or skip winding, spiral chute or vertical conveyor. The staple shaft can be used also as a waste bunker and to accommodate the pipe-line for waste.

Section 2. Cage Winding

The system used in staple shafts is similar to main-shaft winding and only differs in its smaller handling capacity, lower speed and the use of either electricity or compressed air as the motive power.

These systems can be considered under the following sub-headings:

(*a*) *The cages* can be either single- or double-decked, each deck taking either one or two cars, so that the number of cars per cage varies from one to four. Where only two cars are being wound, the cage is usually single-decked in order to eliminate time wasted in deck-changing. Double-deck winding will be used only in narrow shafts and where the staple shaft is deep, because then deck-changing takes up only a small portion of the winding cycle. The cages are of steel-lattice frame construction, the head and bottom framework being connected with vertical supports and cross-braced, and the intermediate deck, if any, being also connected to the main frame. The sides are usually made from perforated plate, the front and back being open and fitted with doors when used for man-riding purposes. The cage is suspended from a central kingpin, or with low cage construction from corner legs with four chains. The latter method has the advantage in preventing tilting of the cage when decking tubs. Steel is used for the cage construction, light alloys still being too expensive for the shorter staple shaft.

(*b*) *Decking* is usually by hand except in large-capacity staple shafts, where mechanical devices, such as rams and tilting platforms, are used due to the need for accelerated decking times. With these mechanical devices the decking of two cars can be carried out within 5 to 6 seconds, whereas with hand decking from 15 to 20 seconds are required. The installation does not differ from main-shaft equipment, which is discussed in Part V, Section 3.

Double- or single-cage winding can be used, the former giving a greater efficiency if four cars are wound on the lifting two decks.

Double-cage winding has the disadvantage that, without special devices, it can be operated only from one level. This normally applies in flat formations where large outputs have to be wound. The system is predominant in flat-seam working, where cage winding is installed, for both coal and waste transport. Single-cage winding with a balance weight has the disadvantage of lower capacity, but is adaptable to intermediate level winding. Frequently it is necessary to have intermediate level hoisting in steep seam working, and it is preferred where the output capacity is not high. Where the winding capacity is insufficient to meet requirements, double-cage winding with double-drum clutch hoists must be used, enabling intermediate level hoisting to be carried out. Where spiral chutes are in use for coal transport, the single-deck hoisting system can be employed, the cage serving for hoisting materials and men.

(*c*) *Staple-shaft winding hoists* will usually have electrical or compressed-air drive. The electrically operated hoist has a higher initial cost, but the running cost is much less than for a compressed-air drive. Where a staple shaft is equipped with a spiral chute for lowering the coal, a compressed-air engine is more economical if the quantity of material to be raised is limited. The electrical drive is preferred where the duty required is more than 50 h.p. The motor can be either a polyphase slip-ring induction motor, with a variable resistance in the rotor circuit for speed control, or the cheaper squirrel-cage motor equipped with a flexible coupling and geared with epicyclic (planet) or turbine gears. The motor will be run on the medium pressure supply of from 550 to 650 volts, but can be operated at high-pressure voltage with higher efficiencies. The speed regulation with the slip-ring motor can be operated through a variable step resistance such as a drum controller with a lever or hand-wheel. The metallic resistance control requires greater room for operating larger winders, and liquid type controllers are preferred.

The braking can be carried out using manually operated brakes or a regenerative or counter-current system, together with mechanically operated brakes, the traction brake being usually the dead-weight post-mounted type, working on a brake path on the driving pulley or the drums. The disadvantages of mechanical braking are the high degree of wear on the brake linings and the high temperature involved. There is a certain degree of uncertainty on the part of the winding engineman, who has to depend on the winding speed

indicator. An emergency brake, which is brought into operation at 15 per cent. over the maximum speed, is recommended as a precautionary measure. The emergency brake or over-speed release can be operated by a centrifugal power switch and a limit switch in the shaft acting on the releasing magnet of the emergency brakes.

For reasons of economy and security a staple-shaft winder working on a regenerative braking system should be provided with an automatic short-circuiting device on the rotor circuit, since this gives complete security against over-speed and facilitates complete control. The wear on the brake path is reduced as it is only used for slowing the cage movement, and the regenerative system puts power back into the line during the short-circuit period.

By comparison, counter-current braking has the disadvantage of causing strong current impulses in the rotor, and therefore the rotor winding must be insulated if this system is used.

The electrical drive is especially suitable for the incorporation of safety devices which effect a quick stoppage of the engine. They can be installed in the circuit of the circuit-breaker at the master switch and in the brake magnet circuit. The latter should include:

(i) An over-wind switch on the depth indicator, breaking the magnet circuit of the emergency brake in the event of an over-wind.

(ii) An over-wind relay in the shaft, which is operated by the cage when it runs too far and works in the same manner as the over-wind switch at the depth indicator.

(iii) A centrifugal switch coupled with the motor shaft which breaks the magnetic circuit if the speed when banking out the cage is too high. An emergency switch in the motor circuit is provided to prevent an overload on the motor when the emergency brake is operated. An interrupter switch can be placed in the no-volt circuit to break the motor circuit.

Compressed-air hoists are usually operated by means of the piston-type drive; in rare cases the turbine motor is used. The compressed-air piston type, is usually double reversing, with twin cylinders and two cranks set 90 degrees apart. With smaller engines, reversal valves are used, while the larger engines employ a Stephenson link reversing gear. The air consumption is about 2,000 cubic feet per minute in both cases. Due to their slow speed of operation, even when inadequately maintained, they are reliable and have a long working life. The motor crank is connected to the drum shaft

or pulley through a clutch and gear (ratio 1 : 5). The brake-rim can be placed on either the driving pulley or the drums.

For intermediate duties, block motors with vertical cylinders (diesel-motor type) can be used with the piston driving the crank shaft through a connecting rod. The engine is single-acting and has a relatively high speed of between 300 r.p.m. and 700 r.p.m. A double-reduction gear is required, giving a 25 : 1 reduction. As the pistons are single-acting, the inlet air is not cooled by the exhaust air and freezing is avoided.

The air turbine motor may be used for staple-shaft hoists up to 100 h.p. and for speeds of from 700 to 3,000 r.p.m. These motors require high maintenance and their use has so far been limited. The motors are mostly double-helical type air turbines, which start up on non-expansion working, thus giving a higher starting torque and expansion introduced as the motor speed is increased. The air consumption of 1,000 cubic feet per minute is very economical with this class of motor. Wear of the rotors will increase the air consumption to 1,200 cubic feet per minute, which is equivalent to that of the slow-running, double-acting, twin-piston type drive.

The hoist construction is very similar whichever form of drive is used, either electrical or compressed air, since the geared motors require the same size of bedplate. The slow-running twin-piston drive takes no greater space, as the motor space is compensated for by the need for only small gears.

(*d*) *Signalling apparatus* is usually the normal balanced hammer and bell arrangement, or a compressed-air-operated bell device. Besides the normal onsetters' signalling arrangement being used for the winding of men and materials, a hammer, to be used in the opposite direction, should be installed, by which repeating of indistinct or confused signals may be requested. A special signalling device for the use of shaft men is often included in signalling equipment, the pulling wire being inserted in a tube to prevent accidental handling. Electrical signalling apparatus is usually installed in the larger capacity shafts and its use is being extended to cover all staple-shaft winding. This type of signalling increases the safety of the system as well as reducing the overall winding time. A speaking-tube or telephone also should be installed for level communication, especially in the deeper shafts.

The signalling apparatus to be provided under the C.M.A. 1911

should be capable of both audible and visual signals, according to the code specified by regulation, with any additional signals required by the manager.

Section 3. Skip Winding

In the Ruhr, skip winding is not so extensively used as cage winding or spiral chutes for staple-shaft transport.

In the first instance, the system is more applicable to hoisting from a lower level to an upper level than for downward transportation, where the use of the spiral chute is better. Where there is a

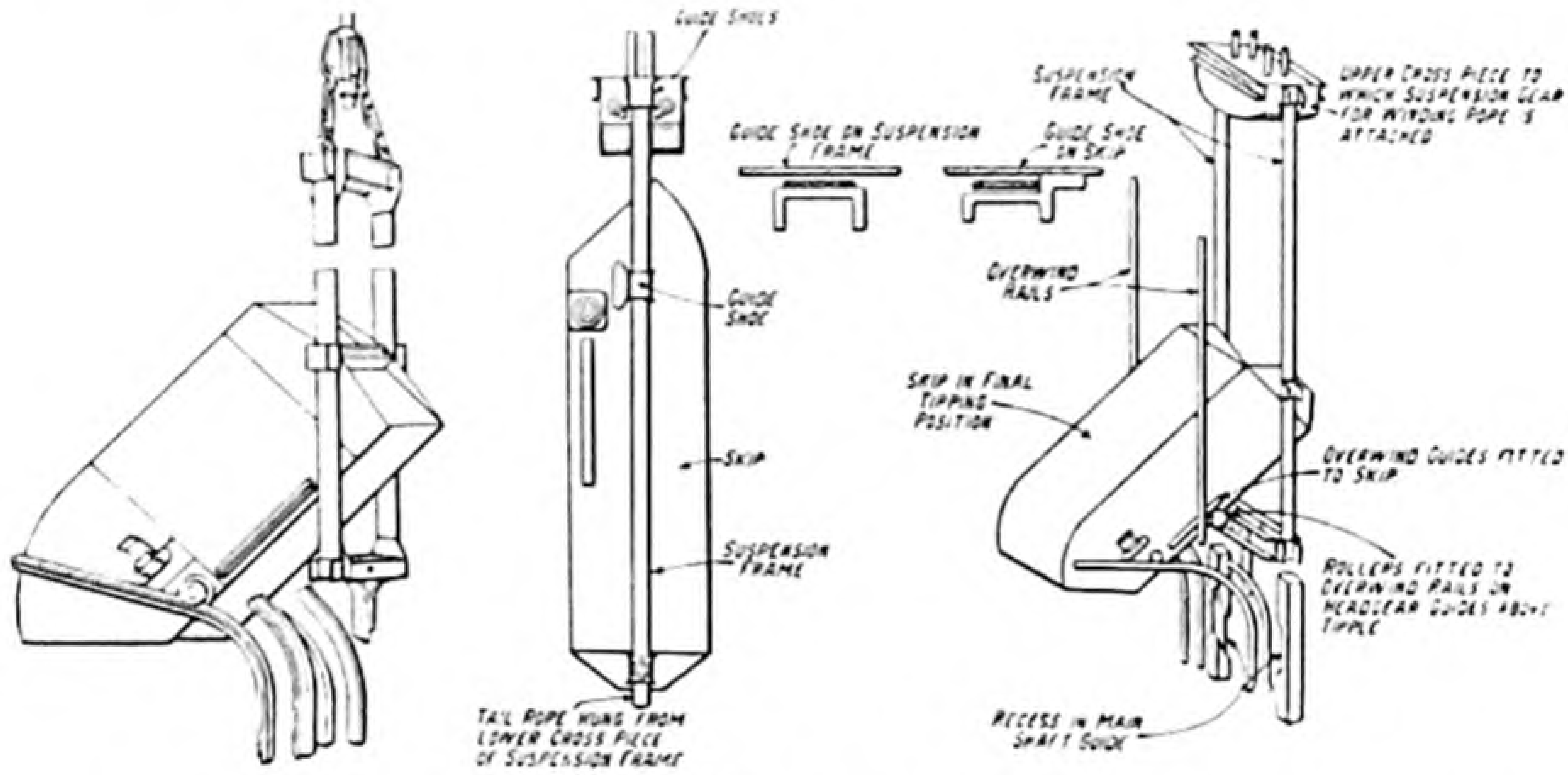


FIG. 257A

possibility of dip-workings being developed below a main haulage level, skip winding may be applied. Skip winding is more widely used for hoisting waste material to the upper return air level or to gate roads, where the normal means of transport is by belt conveyor and not by mine cars, or where pneumatic stowage is being used and delivery of the waste material is arranged to a hopper serving the pipe-line to the face.

(*a*) *Construction of skips.* The skips may be of the overturning or bottom-discharge type, the former type being illustrated in Fig. 257A.

The overturning skip has a simple construction, but has the disadvantage that greater force is required to overturn and dump the gross load. Since the winding rope is slack when the dumping operation occurs, only drum winding should be used with this type of skip; if Koepe winding were used, there would be the danger of rope slip. The length of the overturning skip in relation to its cross-

section is also limited by reason of its form and the shock forces incurred during the dumping operation requiring a larger cross-section of skip than is the case for a bottom-discharge type. In order to minimise shock effects, the overturning skip must be hoisted slowly into the shaft discharge station, so reducing the winding capacity by increasing the actual winding time per skip.

The bottom-discharge skip, as shown in Fig. 257B, avoids these disadvantages and permits a more careful handling of the coal, and is favoured for both coal and waste hoisting. The cross-section is rectangular and the dimensions can be arranged to suit the existing shaft cross-section. To facilitate discharge, the skip tapers towards the bottom and this feature also assists in quickly providing a cushion during loading, thus minimising breakage of the remaining coal charge. To reduce the time for emptying, the mouth of the discharge section is as large as possible and the design of the shutter is very important. The discharge system must prevent lateral flow of the material while emptying, and should be quickly and easily operated and self-locking. The type shown in Fig. 257B fulfils these requirements. The shutter is operated by crank levers which are controlled by the movement of the guide rollers entering the curved guides in the shaft at the transfer or discharge point. Discharge movement in the skip commences as soon as the guide rollers begin to move the cranks, thus giving a partial release of the shutter and starting the discharge before the skip reaches the final transfer position. On the downward movement of the skip, the closure of the shutter is automatic as soon as the skip begins to move. The crank and hinge shafts are beyond the dead-centre in the closed position and the weight of the door and the material assist in keeping the shutter closed. Additional precautions are usually taken to retain the shutter door in the closed position during winding by incorporating spring lock devices in the hinge shafts. If several filling or

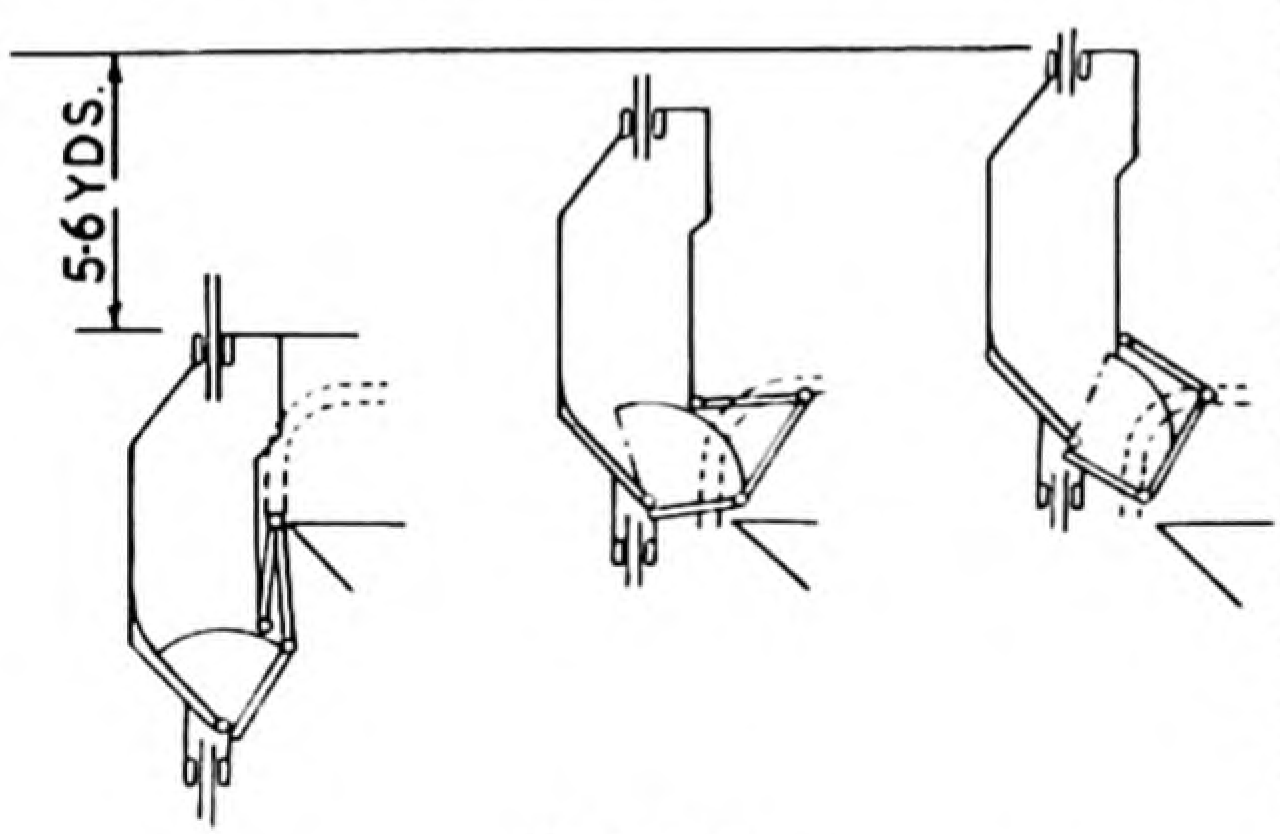


FIG. 257B

discharge stations have to be served by the same skip, the curved guides at the different shaft stations can be switched out of commission as required. The application of skips for man-riding or material transport is frequently carried out in staple shafts, even though the skip requires a false bottom, which makes it a rather heavier and longer skip than that used for coal or waste transport alone. If this disadvantage is to be avoided, it is necessary to provide cage winding in a neighbouring staple shaft as alternative transport for men and materials. The skips are usually made to accommodate a pay load of from 2 to 4 tons, and the weight of the empty skip is from 1 to 1·2 the pay load it carries.

(*b*) *Filling and discharging the skips.* The loading of the skip is carried out from filling pockets or hoppers, into which the coal or waste has already been loaded from a belt conveyor or mine cars. The size of the filling pockets is generally designed by taking into consideration the capacity of the skip and to eliminate large bunker capacities, which are too expensive and require special division arrangements for measuring out each skip load. These may lead to unnecessary coal breakage and difficulty in discharge for a comparatively small installation. The filling pocket or hopper outlet is normally closed by a swinging door operated by means of a crank-lever arrangement. Opening and closing is easily and quickly done, so that delays in hoisting do not occur. The discharge time for a pay load of 2·2 tons should be roughly 3·5 seconds. A fully automatic skip shutter for normal hoisting is shown in Fig. 258. A fork device operated through the side wall of the pocket engages a roller attached to the skip. The downward movement of the skip lowers the shutter, opening the pocket to discharge into the top of the skip. The reverse occurs when the skip is raised, closing the filling pocket shutter, which is counter-weighted to prevent accidental opening.

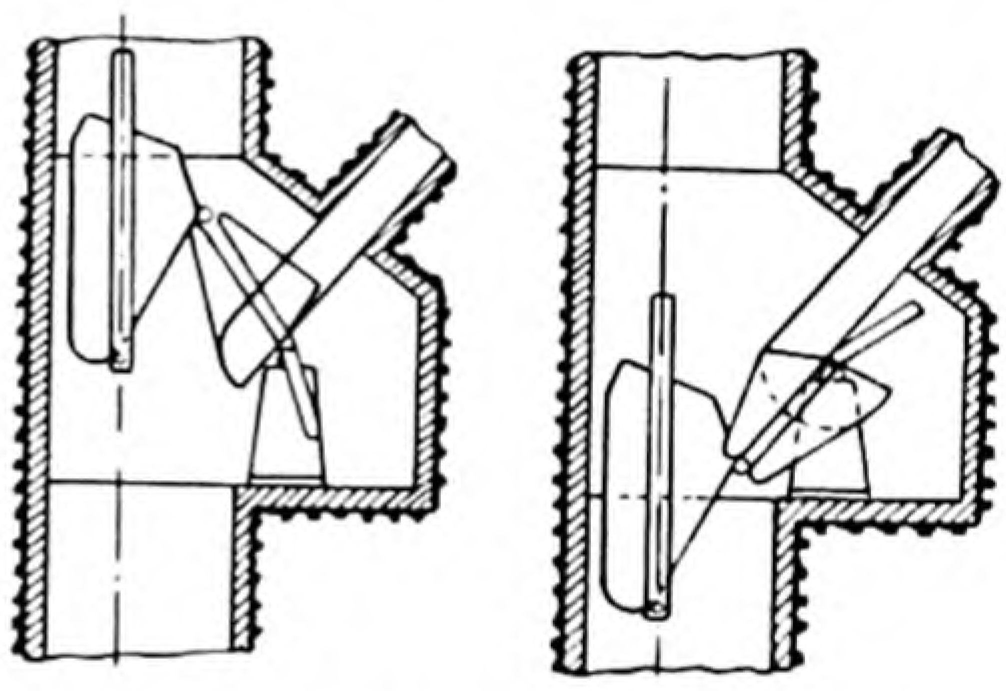


FIG. 258

In the case of two-skip hoisting with a double-drum winder, two filling pockets, side by side, are used as shown in Fig. 259. The

pockets have a common hopper leading into the chutes of the pockets, the outlet into the chutes being electrically controlled through a switch-operated shutter. This shutter closes one or other of the chutes from the hopper above, depending upon which skip is being loaded. A similar hopper and chute arrangement as in Fig. 260 may be used in single-skip hoisting, in which case the hopper is being filled during the hoisting of the skip to the upper level. With coal winding, it is often important to reduce degradation. Special devices attached to the large skips used in main-shaft winding are seldom used in the small skips for stable-shaft winding. Sliding tables, which are brought into their initial position by a pneumatic cylinder and gradually sink under the pressure of the delivered coal, are often fitted in the filling hoppers or chutes. The discharge of the skip is brought into bunkers provided above the main haulage road or sub-level as in Fig. 261. These hoppers usually have a capacity of from one and a half to twice the coal-carrying capacity of the skip. When hoisting waste, these bunkers are frequently made larger, since a certain bunker capacity is required for waste stowage, in which case the bunker delivers to the pneumatic stowage machine. Where the next haulage stage is by mine cars or conveyors, a side outlet is desirable to facilitate easy loading, as shown in Fig. 262. The avoidance of spillage during the filling or discharging operation is not always possible, and to assist in eliminating the trouble of frequent cleaning up, the device shown in Fig. 263 may be incorporated in the loading arrangements.

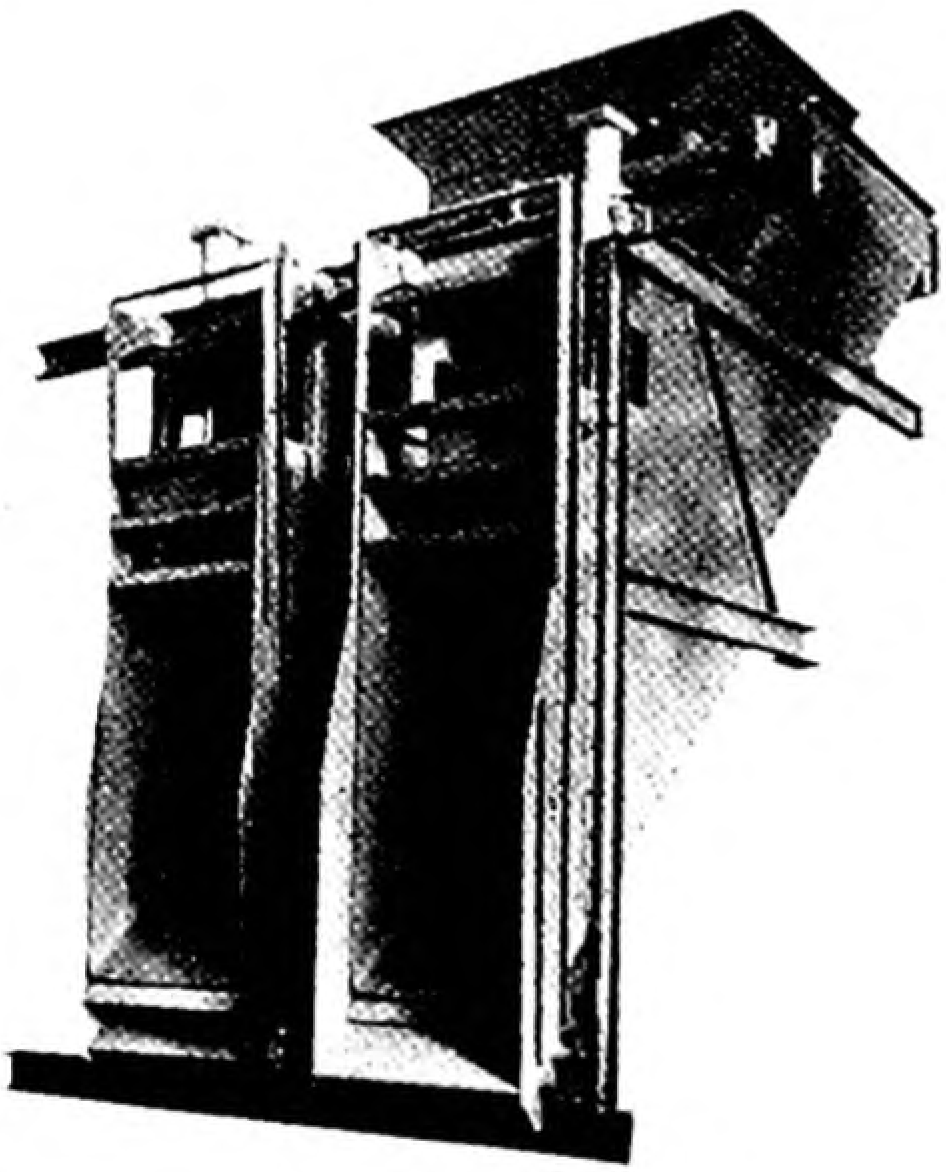


FIG. 259

The question of end-on or side filling and discharging is important. The side is more favourable, since there is more space available and a quicker filling and discharging time can result. The use of side operation is applicable only where single-track haulage is used, whereas with double-track haulage end-on filling is necessary.

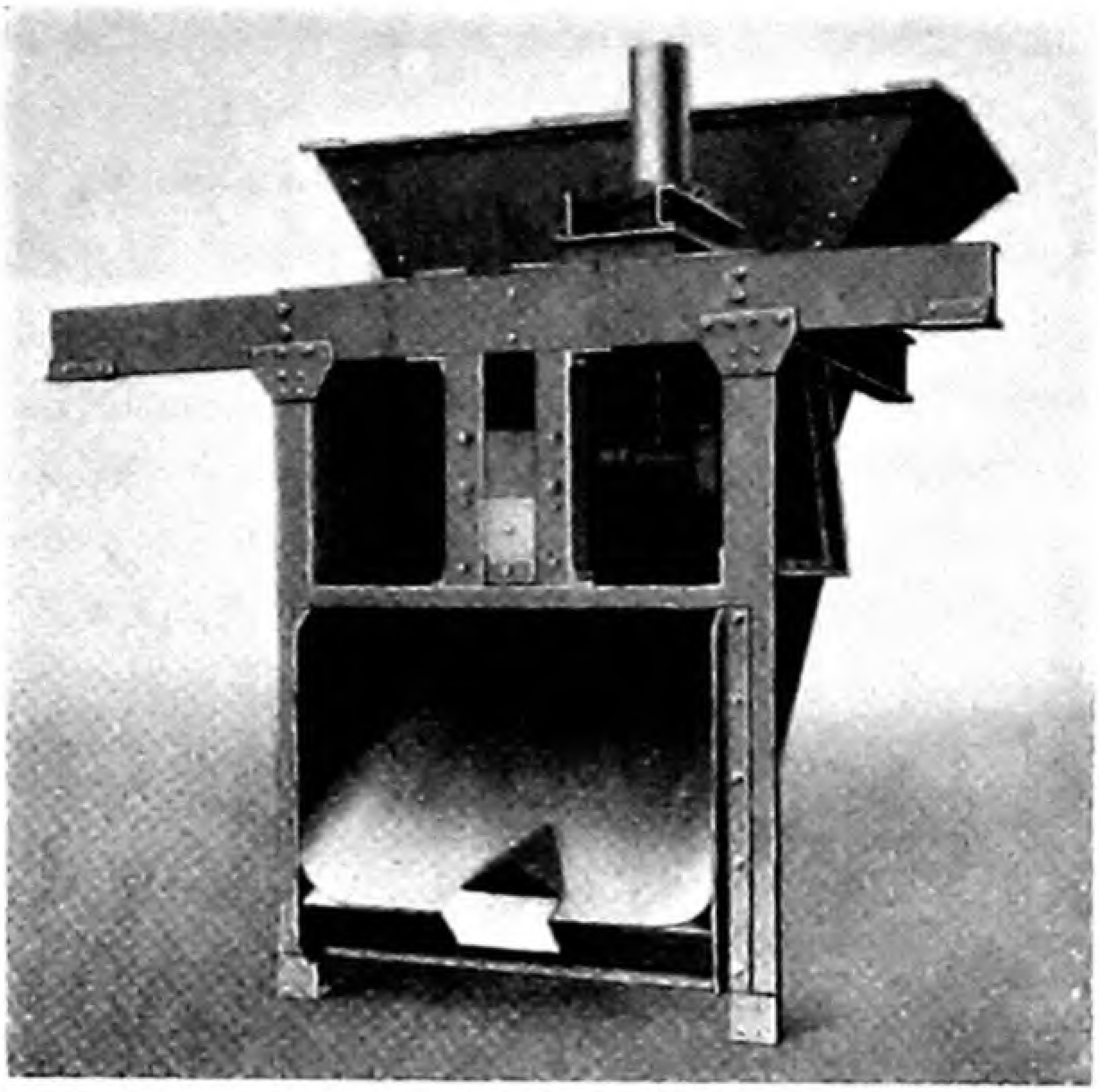


FIG. 260

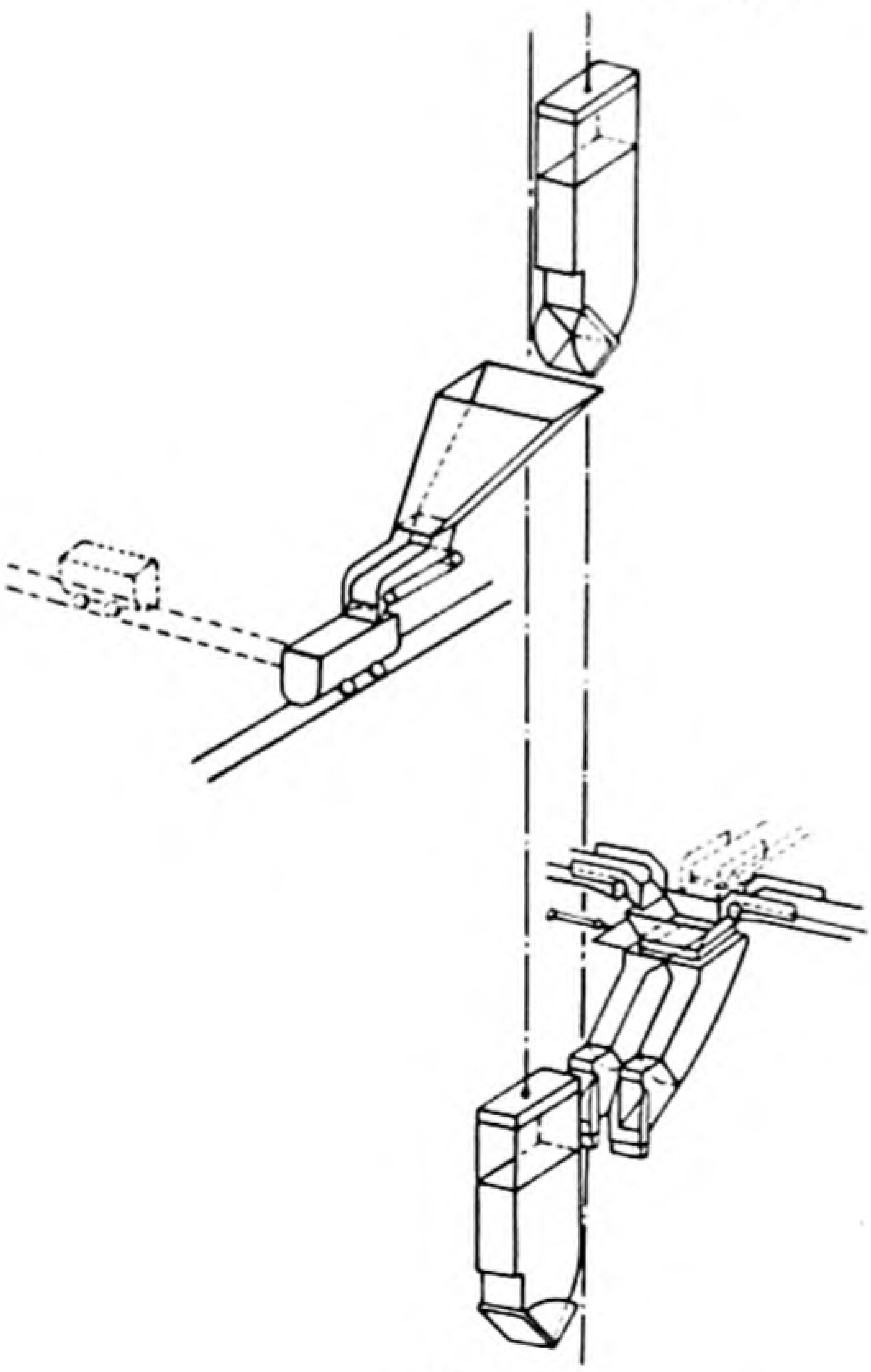


FIG. 261

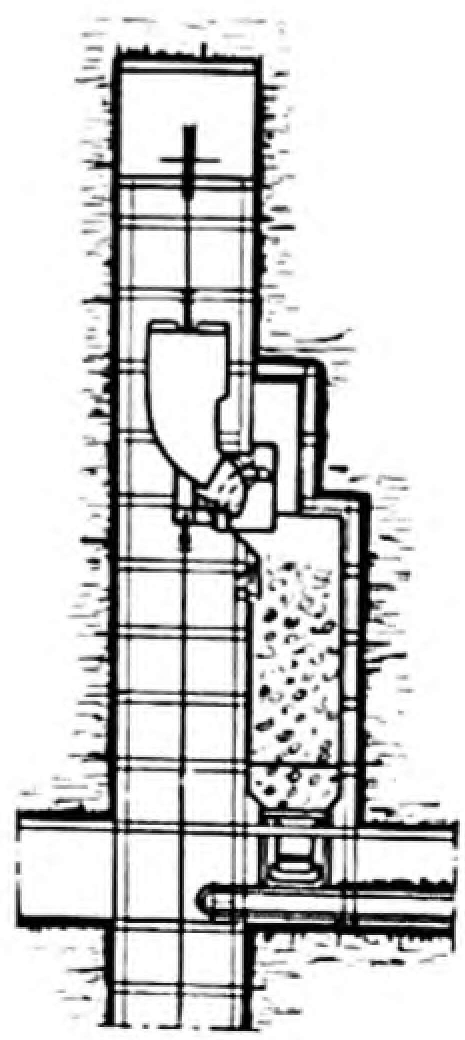


FIG. 262

Section 4. Comparison between Skip and Cage Winding in Staple Shafts

The essential advantages of skip winding over cage winding lie in the reduction of waiting time between winds, and the possibility of using skips of fairly high capacity and small cross-sectional area so that they take up less room in the shaft. These factors all contri-

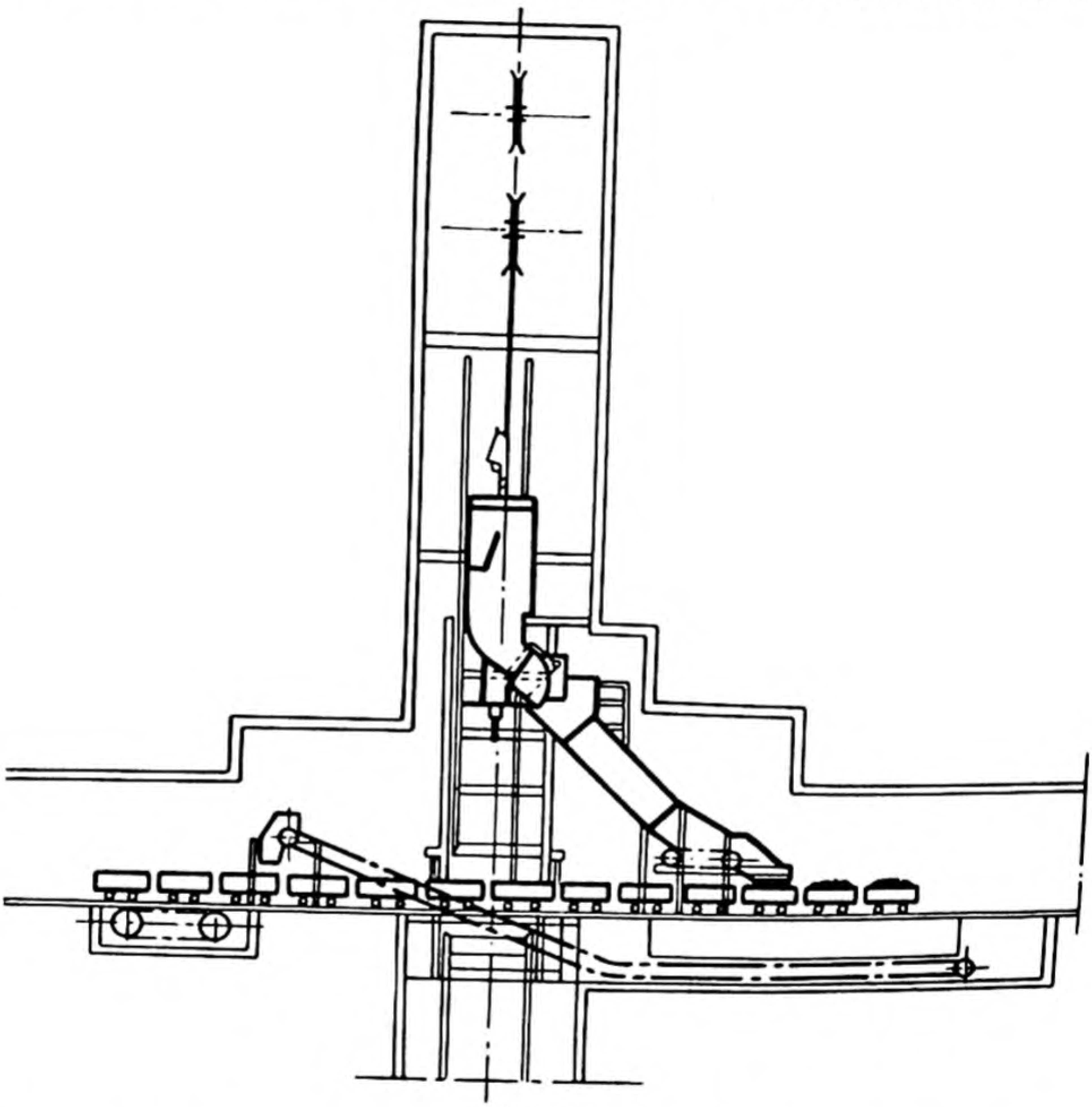


FIG. 263

bute to the greater winding capacity normally achieved with skip winding. The increase in capacity effected by the reduction in the waiting periods varies from 20 to 35 per cent. more winds per shift than under similar conditions with cage winding. This saving will, of course, decrease with any increase in the shaft depth, since the proportion of total winding time taken up by waiting periods will be reduced. On the other hand, when skip winding in shallow

shafts, an increase in the length of wind of about 30 to 40 feet would have a noticeable influence on the total winding time, since the skip must be loaded below the shaft station and discharged above the main-level station. An additional advantage is the more favourable ratio of useful, or pay load, to dead load, due to the fact that the elimination of mine cars or tubs in skip winding reduces the gross load carried for the same pay load. The reduced gross load allows the use of a rope of smaller diameter and weight and a smaller winding engine, and also reduces energy consumption. While these advantages may make an initial skip-winding installation preferable, a change from cage to skip winding can be made, using the same winding engine, which increases the pay load and the original winding capacity considerably. The graph shown in Fig. 264 demonstrates the winding capacity obtainable in skip-winding installations at different depths and winding speeds, when the useful load is 2, 3 or 4 tons. The reduction in the wages cost may be as much as 5 : 3 in favour of skip winding, since the number of men employed at the filling and discharge stations is less than in cage winding. A reduction in the rolling stock required for handling the output raised is also effected. The main disadvantages of skip winding are the modifications required for man-riding and materials transport and its general unsuitability for purposes other than coal or waste hoisting. Cage winding secures a more careful handling of the coal with less liability of breakage, although anti-breakage devices are incorporated in main-shaft hoisting. The simultaneous winding of coal and stone in the same or opposite skips cannot be effectively

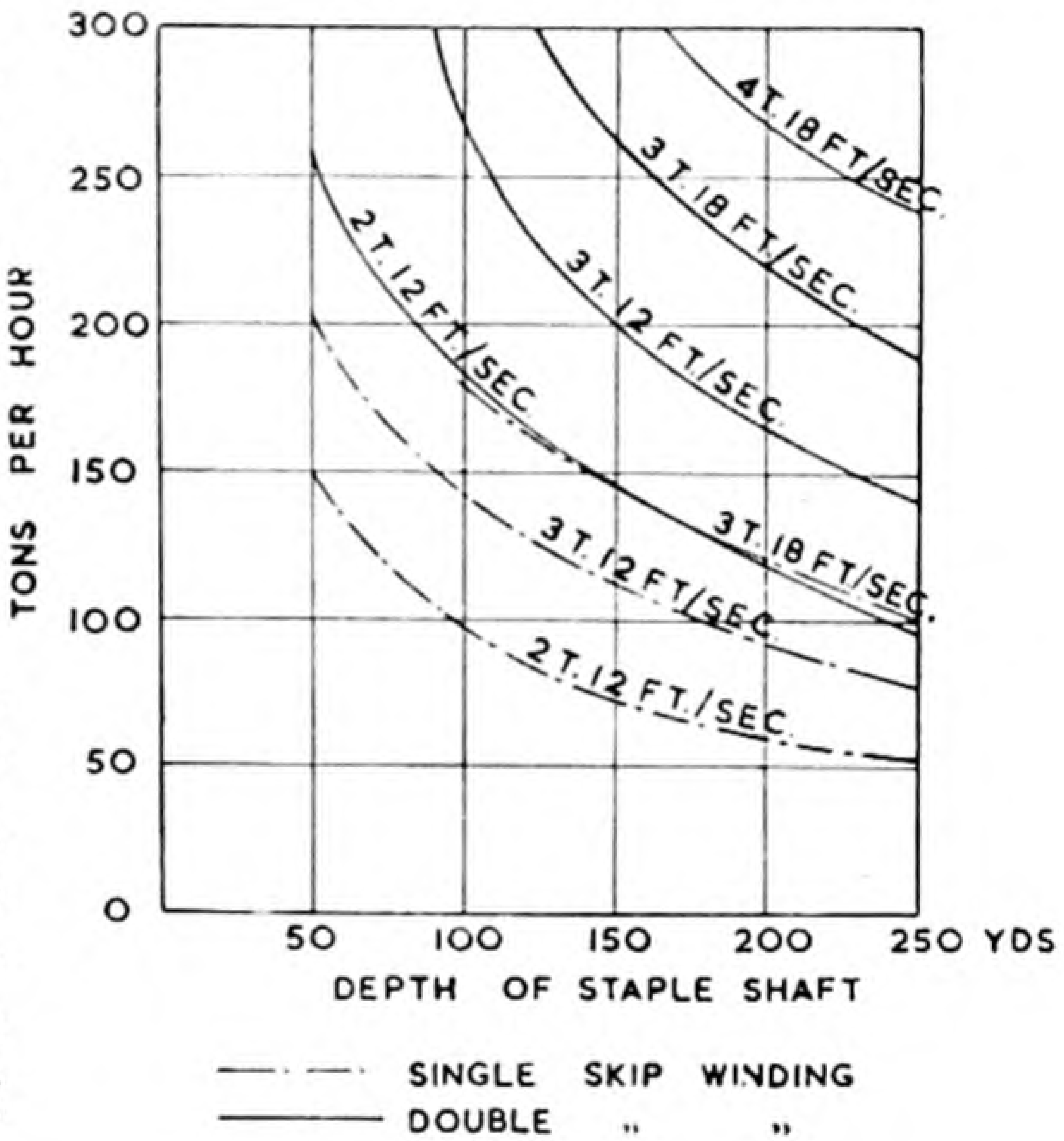


FIG. 264

carried out in skip winding. In these cases it is simpler, and has been carried out in a few cases in the Ruhr, to adopt a cage and skip-winding system with one in each compartment of the shaft.

Section 5. The Drum and Koepe Systems of Winding

As in main-shaft winding, either the drum-type winder or Koepe winder can be used for staple shafts. These are described asfollows:

(*a*) *Drum winding*. A drum and winding rope are required for each cage. The drum cheeks are made of cast iron, while the drum is made of steel and lagged with wood. With certain drum diameters, the width of the drum is decided by the rope diameter and the number of coils required on the drum. The rope is usually connected to the drum through a hole in the rim and fixed with clamps on one of the spokes of the drum cheek. The rope is led from the drums over the rope pulleys, which are mounted in the same vertical plane if the axis of the winding drum is parallel to the direction of car entry into the cages, or side by side if the drum axis is at right angles to the car entry direction.

The lateral deviation of the rope between the drum and the pulley, or 'fleet' angle, should not exceed 1½ degrees. To keep within this limit there should be a certain minimum distance between the drum and the pulley, being greater the wider the face of the drum. The pulley position also has an influence on the size of the fleet angle. Where the pulleys are set side by side, the angle is less than in the vertical-plane mounting, the former therefore being preferred. With deep shafts, the weight of the rope becomes important. The out-of-balance rope weight can be compensated by using tail ropes. With this system, however, winding from intermediate levels is precluded due to the balance rope, and the system is applied only where winding is done from one level. Balanced winding with tail ropes is adopted only with parallel-drum winding. Where the drums used are conical or cylindro-conical drums, balance is achieved by operating on equivalent turning moments on the drum shaft. These drums are, however, large and expensive and are seldom used for staple-shaft winding.

A main disadvantage with drum winders is the space required. In Germany and Holland, only this type of winder is normally used, where double-cage winding is to be carried out at several intermediate levels. In these cases the clutch-type winder will be installed. The

clutch winder is operated, as in main-shaft installations, by holding one drum with the brakes at the upper level, declutching and running the loose cage to the intermediate level and resetting the clutch ready for winding.

(*b*) *Koepe winding* has the main advantage of requiring less space for its installation, is cheaper in its initial and operating costs and more efficient than the drum winder, as the masses to be accelerated and brought to rest are much smaller. The winder can be installed in the staple shaft itself and does not require a separate chamber at the side of the shaft as with the drum winder. The winder can be placed either at the upper or lower level. The installation of the rope is simpler and the system is widely used in both Germany and Holland.

In this system, one cage is suspended at one end of the winding rope with the second cage attached to the same rope at the other end. The rope is driven by the friction drive of the Koepe pulley, and although it was sometimes believed that such a friction drive would be insufficient, thousands of installations, operating successfully for many years, have proved that this opinion was incorrect. The possibility of rope slip can be an advantage in certain circumstances in staple-shaft winding, as in the case of the gauge between the cage guides being reduced, due to strata pressure, and one or other of the cages becoming fast in the shaft. With drum winding, however, such an eventuality would cause a rope breakage.

In normal winding practice, rope slip does not take place, the friction of the Koepe pulley depends upon the frictional grip $e^{\theta\mu}$ where μ is the coefficient of friction and θ is the angle of lap on the driving pulley. The frictional grip can therefore be increased by increasing the coefficient of friction and/or the lap angle. The former method is usually simpler, and special lining materials are employed for the pulley groove. Aluminium alloys having $\mu = 0{\cdot}7$ for dry rope are frequently used, while $\mu = 0{\cdot}35$ is sufficient to provide a positive drive. This alloy material is incombustible. Soft zinc has also been used. Where the shaft is wet, organic resin material will provide a better insert, such as cotton and rubber impregnated with resins. This material has a coefficient of friction of 0·6. Greasing of the rope with light grease of the vaseline type should be avoided, and stiff glutinous greases of the adhesive type should be used. The fastening of the lining inserts in the driving pulley must be carried out carefully. Normally the inserts are dovetailed

and held tightly in position by wedging them in the pulley groove.

When employing a metallic lining, care has to be taken when the block wears down that it is not separated into two parts which may be flung off during winding operations to the danger of the engineman.

Section 6. Cage Guides in Staple Shafts

The same methods in use in main shafts for cage guides, such as wooden or rail fixed guides and rope guides, are used in staple-shaft winding practice. A staple shaft, as distinct from a main shaft, is generally subjected to strata pressure, due to mining operations, and requires guides which will provide the greatest safety even when the verticality of the shaft or the shaft section is altered. In Continental practice, fixed wooden guides have been found to be advisable under these conditions. In this case, the compartment dimensions are chosen to allow sufficient guide clearance even if the guides are affected by strata pressure. It is recommended that wooden pads should be set between the guide shoe and the buntons, and removed after the strata movement has commenced. A special form of guide holder which permits lateral and vertical movement of the guide has proved to be very useful in the Ruhr and is shown in Fig. 265.

FIG. 265

Steel guides have the disadvantage of causing severe shocks to which the cage is subjected even from slight irregularities in guide alignment. In addition to this fact, repairs are more difficult to carry out than with wooden guides where compression of the guides occurs. Rope guides have the disadvantage of being unable to allow for deviations in the verticality of the shaft, which deviations are likely to occur in most shafts after extraction has commenced in the adjacent seams.

Section 7. The Installation of Staple-shaft Winding Engines

In the past, only two methods of installation were adopted for staple-shaft winding engines. In the case of small Koepe winders, these were normally installed in the upper end of the shaft itself where the necessary extra height was obtained by extending the shaft to provide the engine chamber, as shown in Fig. 266(B).

Drum winders or larger Koepe winders, however, were installed in a special chamber alongside the upper landing, or inset, as in Fig. 266(A). In rare cases, the winder may be placed at the side of the bottom inset, when the winding rope must be run over guide pulleys placed in the shaft to the rope pulleys installed at the shaft top. Another method has now been used for several years, in which the Koepe winder is installed beside the lower inset in such a way that the driving pulley is positioned in the ladder-way and the ropes run in this compartment to the guide pulleys, as shown in Fig. 266(C).

The installation of the engine at the top of the shaft has the advantage that the smallest possible space and shortest length of rope are required. Since the rope is only travelling over a driving pulley during each wind, the rope deterioration due to bending stress is reduced and therefore it has a longer life. A disadvantage of this layout is the provision of auxiliary ventilation for such winding chambers when installed at the shaft top in a gassy mine. Where the winding operations are conducted in one compartment only, using a balance weight, the engine must be installed in an oblique position to the shaft cross-section, as shown in Fig. 267. This layout is necessary in order to bring the respective ropes into the centres of the winding and balance-weight compartments.

Side-chamber installation is necessary for all drum winders, since space is required to give the necessary fleet angle between the centre and outside rope positions on the drum and to provide for the distance between the engine and shaft centre. This is also practically the only position possible for large Koepe winders and provides good facilities for supervision, maintenance and ventilation of the engine. A disadvantage of this method of installation is the provision of a side chamber at a suitable distance from the shaft. In the case of a Koepe winder this distance is of extreme importance, since in each wind the rope travels over the driving pulley and the two guide pulleys and therefore is subjected to *reverse* bending stress at

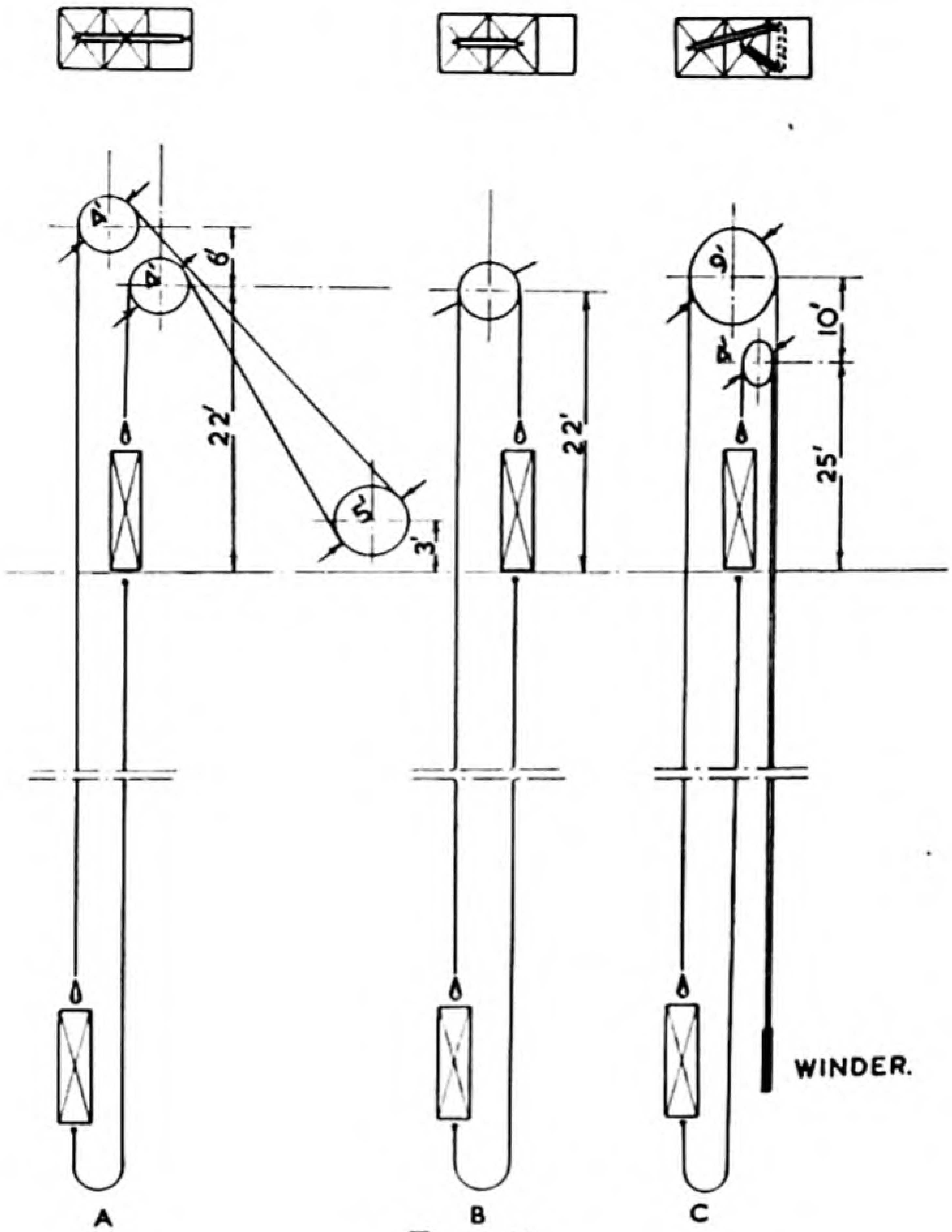


FIG. 266

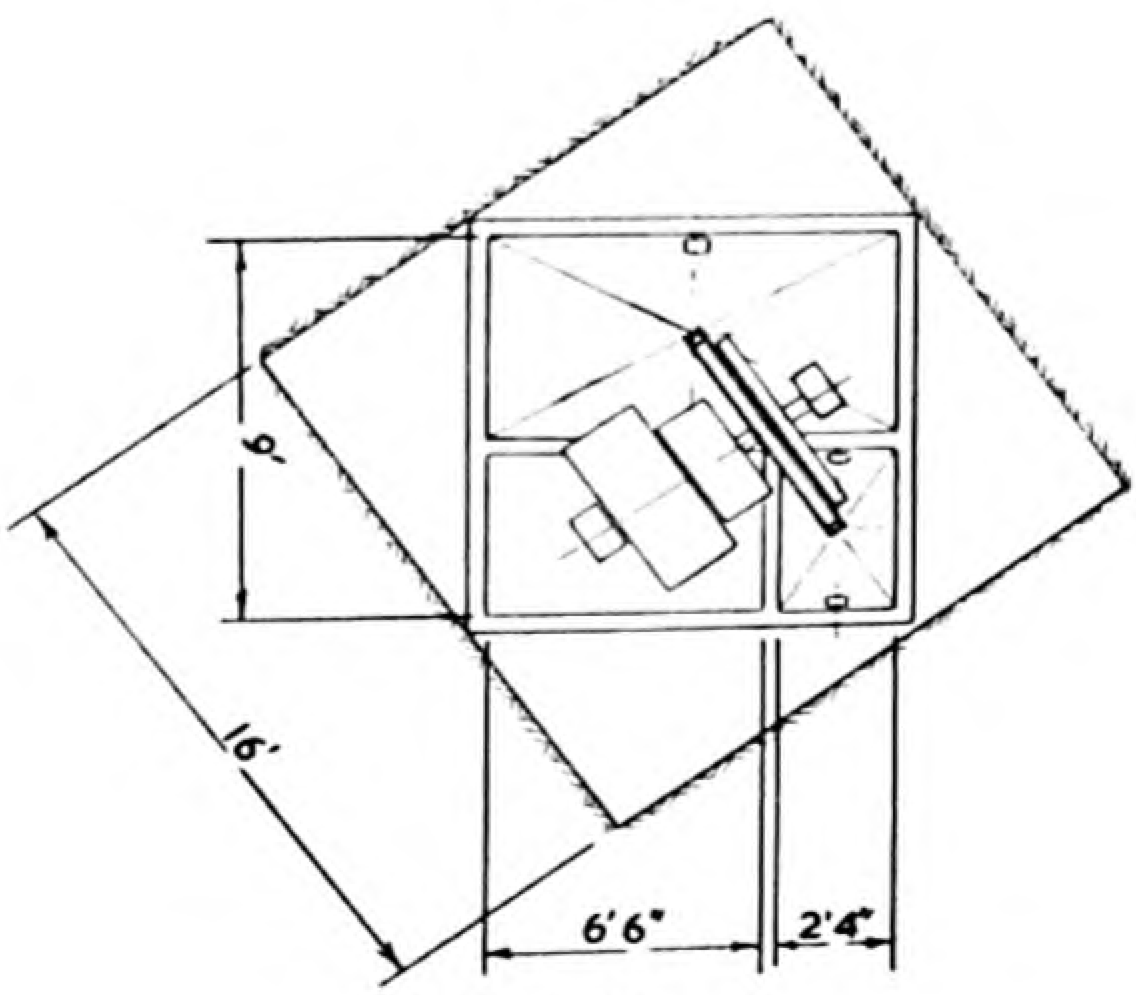


FIG. 267

three points. This means that the rope will have a shorter life than would be the case with an overhead layout.

Shaft-bottom installation is very rarely resorted to where two additional guide pulleys are required. This method has a decided disadvantage due to the high rate of wear of the ropes, even where the guide pulleys are the same diameter as the rope pulleys. Where the drive pulley is situated at the lower end of the ladder-way, eliminating guide pulleys, the arrangement gives easy accessibility, simple ventilation, reliable foundations and eliminates the provision of a special engine chamber and a channel in the roof to accommodate the ropes. The provision of electric cables in the shaft is obviated with the shaft-bottom installation. A disadvantage of this system is that large rope pulleys are required, probably up to 9 feet in diameter, as shown in Fig. 266 (c), the transport of which may be difficult underground. The installation of such large pulleys makes the shaft timbering difficult to insert near the pulley position, which may cause difficulties in broken ground. The rope is vertical and there is no horizontal component of the pull trying to shift the pulleys. The winding rope must be twice as long as the depth of the shaft, whereas with other methods of installation the rope is about the same length as the shaft. In the former case, however, the rope is bent over one pulley only and its life is consequently about twice as long as in the latter case. Taking into account the cost of ropes, etc., and the rope life, the ratio of rope cost in the case of side installation at the shaft top and shaft bottom is about 1·25 : 1 assuming that the shafts are dry and no corrosion effects are present.

The greater length of rope used in shaft-bottom installation has disadvantages, however, due to stretch and the formation of slack when loading and discharging the cage during decking operations. At a depth of 150 yards when unloading a single tub weighing 1 ton loaded, in a single-cage system, it should be anticipated that the rope shortens by about 4 inches. With new ropes, the stretch may be even greater and in certain cases it may be necessary to install tilting platforms. It may be possible to build up the floor level on the full side above the discharging, or empty side, at the inset. This method, of course, precludes the raising of empty tubs or packing material unless special arrangements are made to bypass such tubs to the decking side. Rope maintenance is also more difficult, since

there is no point in the shaft where the complete length of the rope can be examined and two or even three points have to be taken.

In many cases, especially in shafts of shallow or medium depth, the disadvantages are outweighed by the advantages already discussed in respect of the location of the engine and pulleys at the shaft bottom inset.

Section 8. *Winding Ropes*

The type of rope used varies in Koepe and drum-winding systems. The locked coil rope shown in Fig. 268 is usually preferred for drum winding in Britain, while on the Continent the Lang's lay rope and the ordinary lay rope, as shown in Fig. 269, are more widely adopted.

With the locked-coil construction, the periphery of the rope is smooth and, because of the complete interlock provided by the

FIG. 268

trapezoidal form of wires used, the rope cross-section is reduced to the minimum requirement for any specific duty. The construction eliminates any tendency to twist when in use and minimises the danger due to corrosion. On the other hand, internal individual wire failures are difficult to detect and in the course of winding operations the tension effect on the wires varies, with the result that 'bird caging' of the exterior wires may occur.

Ordinary lay ropes are constructed with a counter-lay of the strands (usually six) and the wires composing the strand. The tendency for the rope to untwist is fairly low and the ropes are not inclined to kink. In the case of the Lang's lay rope, this untwisting is liable to occur when 'slack' is formed as the cage is set on the keps. In the Lang's lay rope, the lay of the strands and the wires is in the same direction. This type of rope is more flexible than ordinary lay rope and, due to the slope of the outer wires, offers a greater grip on a Koepe pulley lining and gives an increased friction coefficient for the pulley inserts. In addition to this factor, the outer wires adhere to the base of the rope groove of the pulleys, resulting

in a more even distribution of the rope pressure. The tendency of a Lang's lay rope to untwist can be overcome by adopting the Tru-Lay system of construction, in which the wires of the strands are pre-formed to their final shape before the rope is laid. The ordinary lay and Lang's lay ropes (with six or seven strands) are twisted around a hemp core made of manila or jute. The core is soaked in vaseline or rope varnish to prevent the entry of water and to give internal lubrication. The hemp core forms a soft cushion and prevents heavy pressures on the strands.

(*a*) *Balance ropes.* The adoption of balance ropes equal in length to the depth of the shaft and fastened with the two ends to the

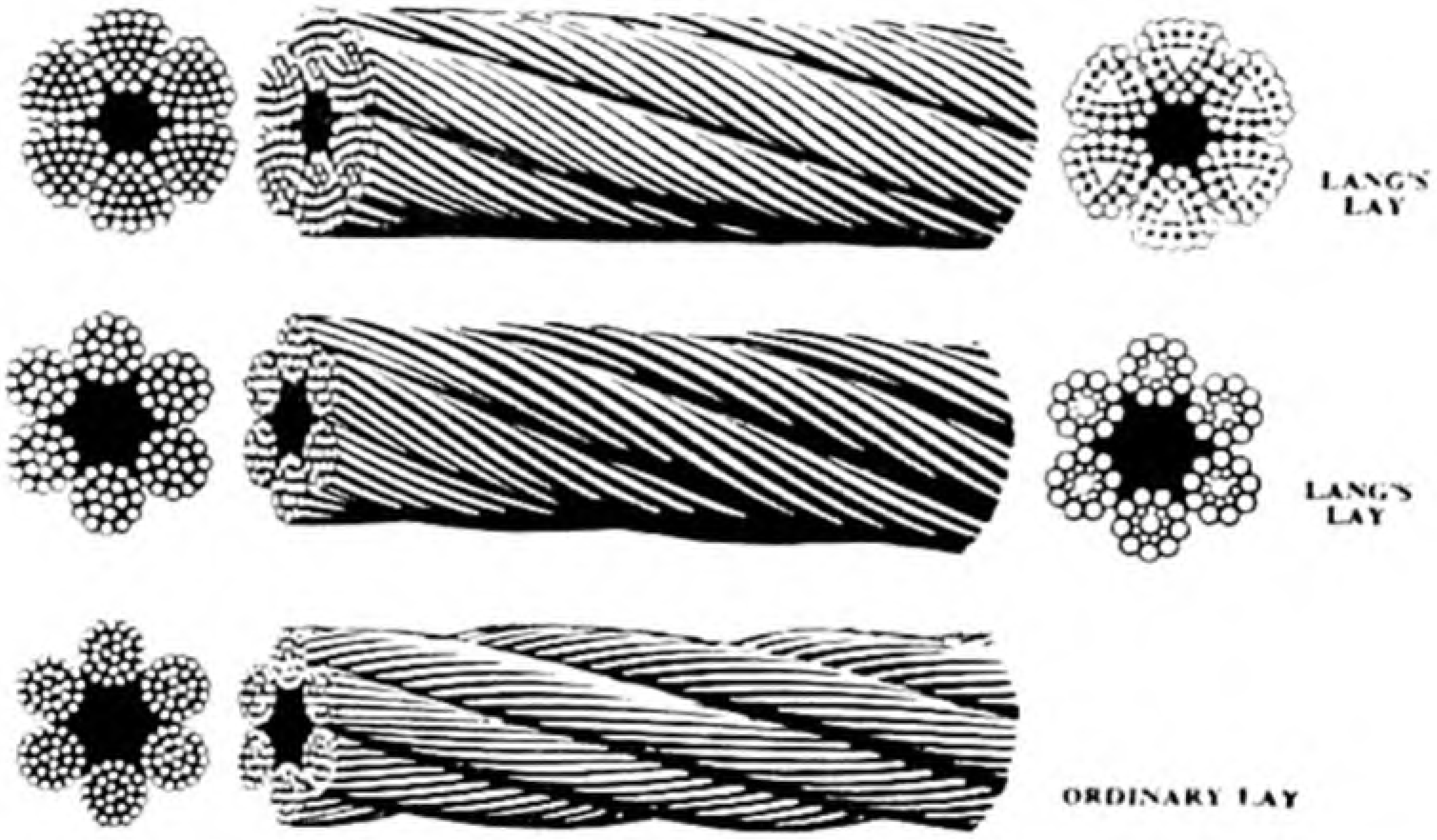


FIG. 269

bottom of the cages in two-cage winding serves to counteract the out-of-balance weight of the winding ropes, thus assisting the winding engine. In Koepe winding, this method is universally adopted. In drum winding, balance ropes are only used when single-level winding is in operation. Ordinary lay ropes are unsuitable for balance ropes because they tend to untwist, so that the wires loosen and are subject to early corrosive effects. Flat ropes are most widely used because of their ability to bend easily and their freedom from twist; one form is shown in Fig. 270.

(*b*) *The testing and examination of winding ropes, chains and other equipment.* In view of the necessity to maintain a high factor of safety in winding, the testing and examination of ropes, chains, rope-capping and hooks are extremely important. In addition to the periodic examinations required while the equipment is in operation.

the examination and testing of the rope near to the capping itself are of great importance, since it is at these points that heavy stress occurs and possible damage is more difficult to observe. In the case of drum winding, this requires that the capel as well as the first 6 feet of rope are cut off every six months. The rope is examined for faults and the capel renewed. Examination of the rope-length cut off should be carried out according to the existing regulations.

With Koepe winding, in which the rope-length used must remain constant, a similar examination of the rope and capel is impossible. After many years of experience in the Ruhr this factor is not considered a major disadvantage, and, instead, careful examinations of the rope must be carried out daily and weekly. The daily examina-

FIG. 270

tion is conducted with the rope running at a slow speed of about 3 feet per second and the weekly one at a speed of 1·5 feet per second. In addition, a close examination is made of the rope every six weeks, when certain parts of the rope are cleaned carefully in order to investigate wear and attack from corrosion. By this method of examination, only the outside condition can be inspected, but experience has proved that the outside condition is decisive in establishing the state of the rope in use. The examination of interior weakness may be conducted with electro-magnetic detectors, such as the magna-flux crack detector or the German Weever-Otto method in which the winding rope is passed through a magnetic field, produced by a series of coils, at a speed of about 0·7 feet per second.

Section 9. Continuous Transport in Staple Shafts

(*a*) *The spiral chute.* In staple shafts, the spiral chute provides a continuous system for the transport of material from an upper to a lower level as distinct from the intermittent transportation by cage or skip winding. Since the chute has no moving parts and requires

no power, its use has been considerably extended. In addition to the low operating cost involved and the high capacity, there is the advantage that the chute loads directly into mine cars in the main haulage level. Thus, the cars are kept at one level, resulting in speedier circulation and a reduction in the number required. Mine-car haulage systems are being replaced by belt conveying for gate-road haulage, which gives continuous coal transport from the face

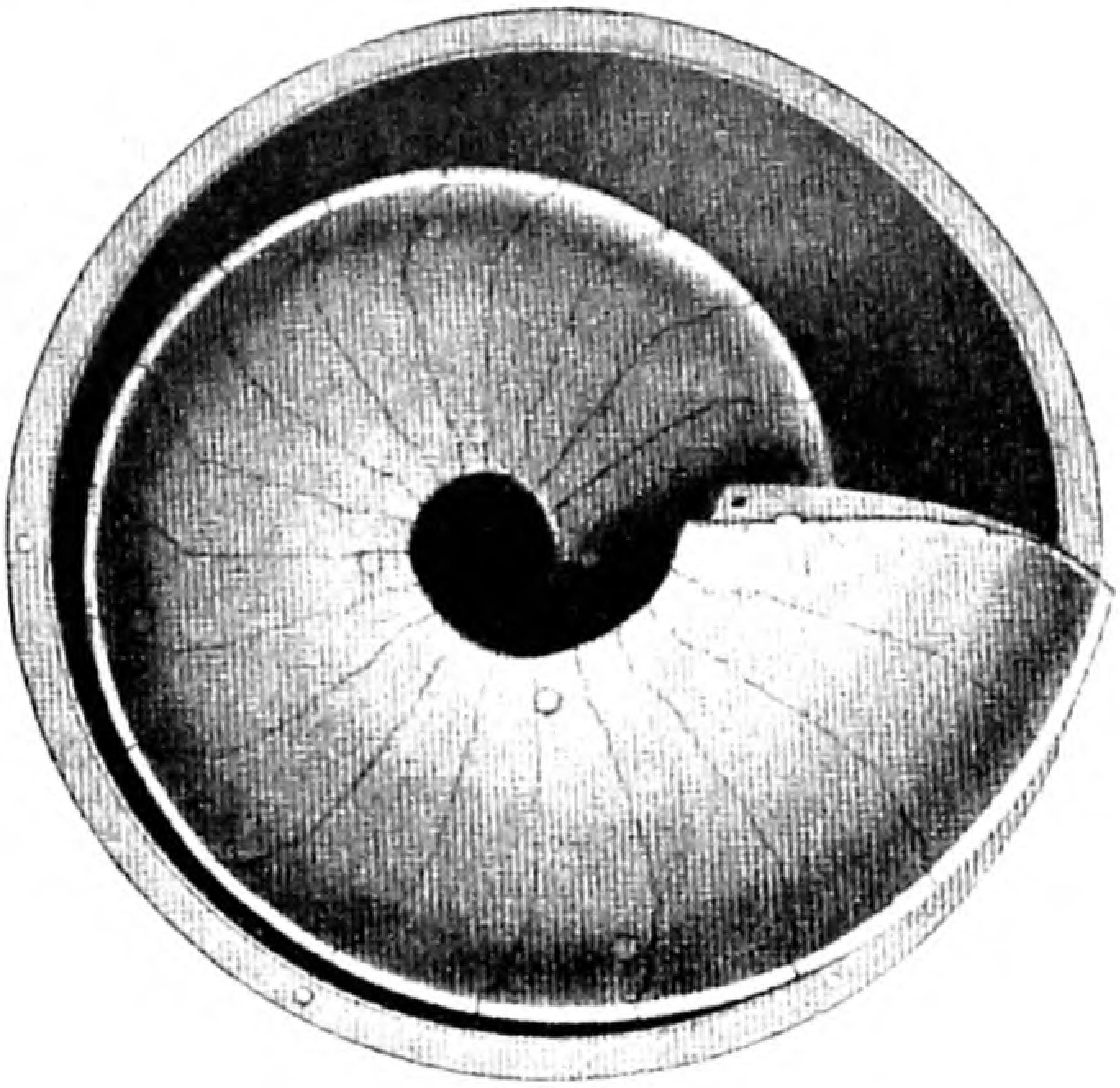


FIG. 271

to the mine car in the haulage level below. In the spiral chute, the coal slides down the spiral track, which is fixed in an outer tube, or casing, as shown in Fig. 271. The spiral chute may consist of one or more inlet sections, the individual spiral sections and an outlet section. In order to facilitate transport and installation of the sections, the height of each section is limited according to the diameter of the chute. The principal dimensions of chutes of British, German and Belgian manufacture are shown in the table on the following page.

The circular outer casing sections consist of mild steel and have flat flanges for connection. The tubes can thus be installed to practically any height. The suspension in the staple shaft can be

Manufacturer	Diameter		Height of Section		Casing	Spiral Plates	Capacity tons/hr.	Size of Coal	Cost per yd. (1949)			Remarks
	ft.	in.	ft.	in.				in.	£	s.	d.	
British (a) .	3	$4\frac{7}{8}$	4	3	$\frac{1}{4}$ in. M.S.	Cast Ni. iron, 4 per pitch	200/250	—12	26	5	0	
,, (b) .	4	0	5	0	$\frac{1}{4}$ in. M.S.	Cast Ni. iron, 6 per pitch	300/350	—18	36	3	4	
,, (c) .	5	0	6	4	$\frac{1}{4}$ in. M.S.	Cast Ni. iron, 8 per pitch	400/450	Run of mine large	42	0	0	
,, (d) .	3	5	2	$1\frac{19}{32}$	$\frac{3}{16}$ in. M.S.	'Firthag' Cr. Mn. rolled-steel plate or carbon steel	—	—	{ 29 20	15 14	0 0	
,, (e) .	4	$10\frac{1}{2}$	3	6	$\frac{1}{4}$ in. M.S.	M.S. $\frac{5}{16}$ in.	500/700	Run of mine large	49	17	5	Including inlet and outlet sections
Belgian (a) .	4	1	2	$5\frac{1}{2}$	$\frac{13}{64}$ in. M.S.	M.S. $\frac{13}{64}$ in., wearing plate chilled C.I. $\frac{15}{32}$–$\frac{19}{32}$ in.	—	—	35	0	0	
,, (b) .	4	1	2	$5\frac{1}{2}$	$\frac{15}{64}$ in. M.S.	Special steel $\frac{5}{16}$ in. (Siemens Martin 40/50 tons/sq. in.)	—	—	30	12	8	
German .	4	1	2	$5\frac{1}{2}$	$\frac{15}{64}$ in. M.S.	Special steel or chilled C.I. $\frac{13}{64}$ in. or $\frac{38}{64}$ in.	500	—	{ 35 37	0 1	0 4	Also made in 3 ft. $5\frac{5}{16}$ in. and 4 ft. 9 in. dia.

N.C.B. Information Bulletin No. MP (49) 3.

carried out in various ways, the usual methods being by fixing the flanges of the tube sections, or by using brackets or plates welded on to the tubes. Examples of various suspension units are shown in Figs. 272 and 273. Special care has to be taken to ensure that a strong support is provided at the base of the chute, since, if the chute is used as a bunker or storage hopper, large loads will have to be carried.

The various manufacturers in Britain and abroad have different designs for the spiral chute; in most cases, the spiral sections are bolted to the inside of the casing and sometimes to each other, so giving a close fit and providing a continuous and constant resistance path to the material handled. The height of each section is usually equivalent to half a pitch, so that two sections are required for a complete spiral through 360 degrees. A typical arrangement of a spiral chute is shown in Fig. 274. The spiral chute should be designed to allow the stream of coal to slide down steadily at a uniform speed. The flow velocity is from 4 to 5 feet per second. It is independent of the height of the chute but varies with the pitch and diameter, and the frictional resistance of the different materials. The tangential angle of the spiral is about 70 degrees at the centre hole and about 23 degrees at the outer casing, these angles varying to some extent with the particular design. With this form of the spiral path, the coal is distributed evenly over the surface of the spiral and blockages are avoided. In addition, inspection doors are provided in each of the chute sections for maintenance and replacement of spiral sections. In order to assist in minimising blockage in the chute, the centre hole is provided, which also increases the capacity. The central hole as shown in Fig. 271 is about 10 inches in diameter and assists in providing an immediate outlet for the fine coal where the chute is being used as a bunker. On the continuation of loading, coal flows from the centre initiating movement in the remaining coal in the chute. The outer periphery of the spiral plates is curved in such a manner as to protect the casing against wear, as shown in Fig. 275. The extent of the wear of the spiral and casing sections depends upon the steel used in their construction and upon the hardness and moisture content of the coal or stowage material being conveyed. The shortest life will be experienced when using steel plates. Where steel of normal strength has been used in the Ruhr, it has been found that quantities of coal ranging from 100,000

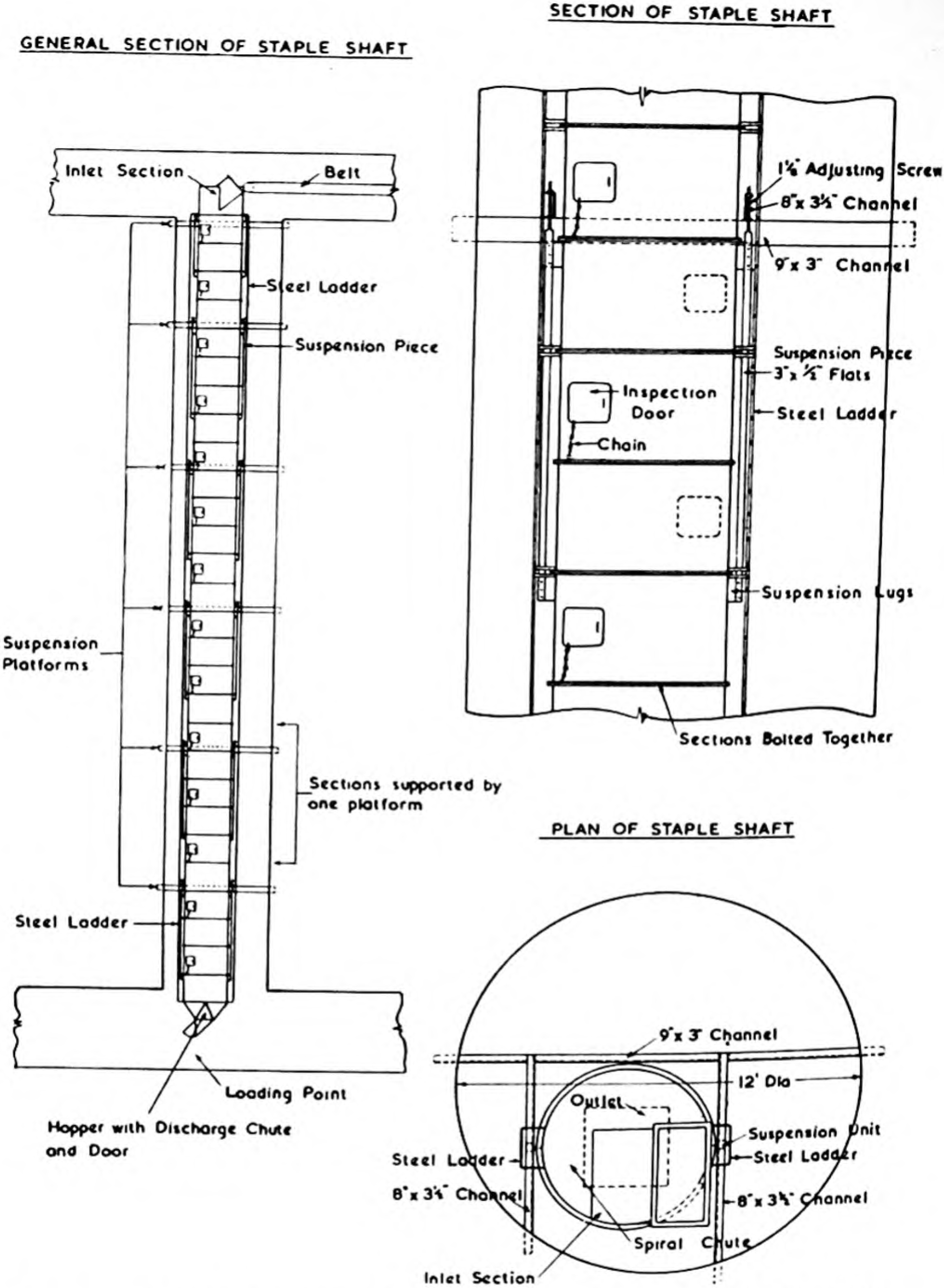
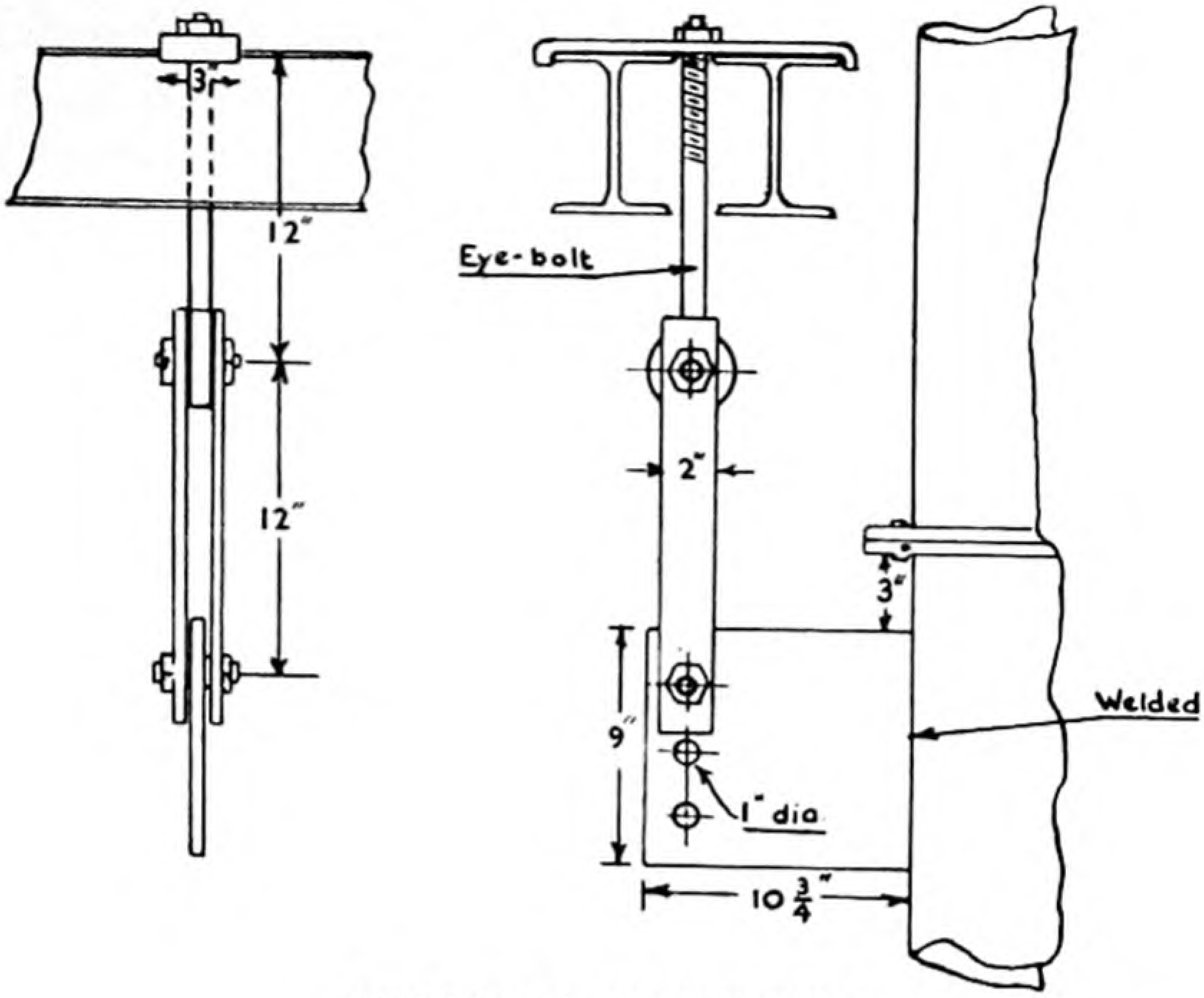


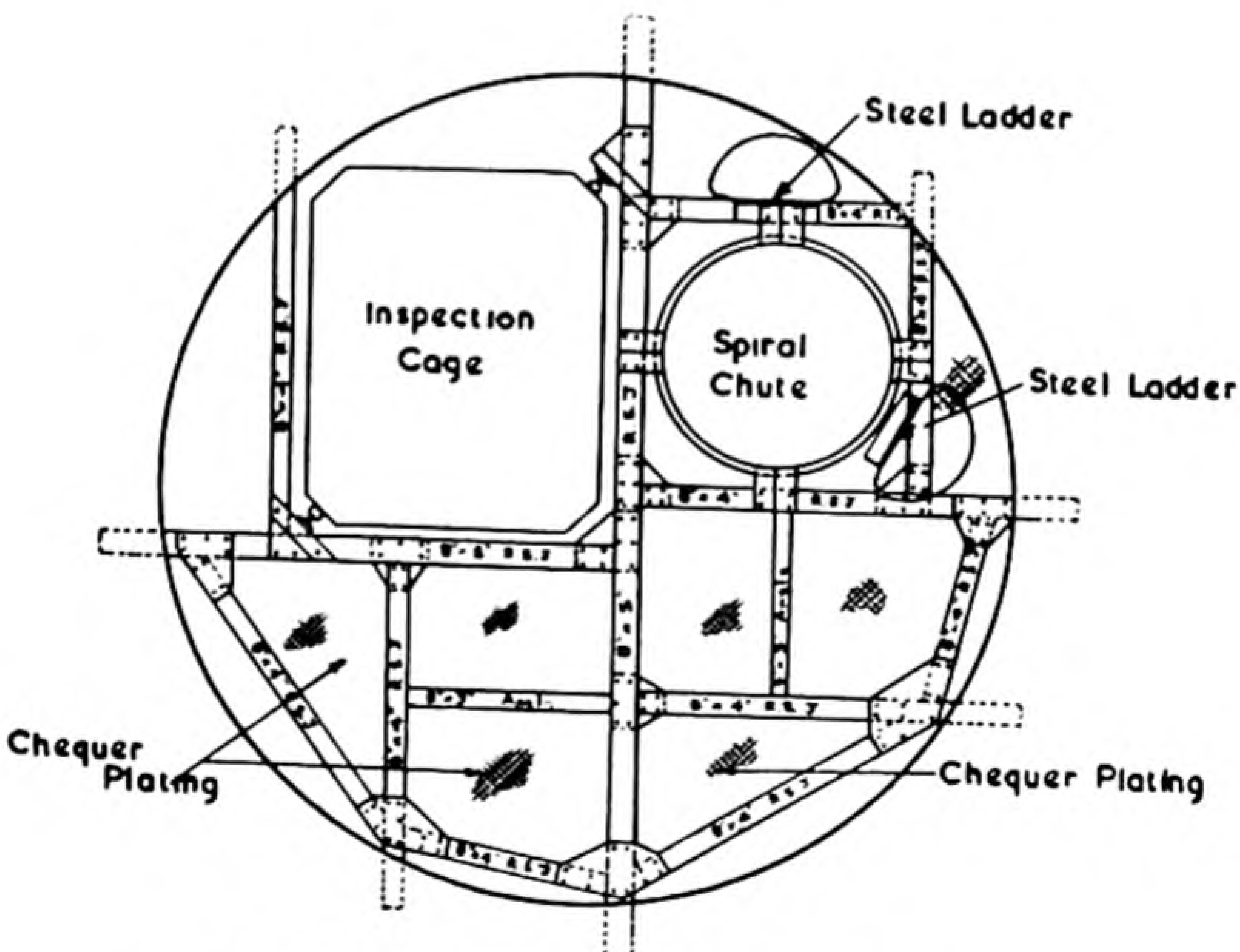
FIG. 272.—Method of support of spiral chute and ladderway at Brookhill Colliery

Details of Spiral Chute.

Manufacturer: Butterley Co. Ltd.
Internal Diameter: 4 ft. 11 in.
Height of Section: 3 ft. $5\frac{3}{8}$ in.
No Centre Column.
Total Length of Spiral Chute excluding Intake and Discharge Sections 96 ft.



SUSPENSION ARRANGEMENT



DETAILS OF SUSPENSION PLATFORM

Fig. 273

to 400,000 tons can be conveyed before plate replacements are required, and so the use of wear-resisting lining on the spiral plates has been adopted. These linings, shown in Fig. 275, are usually segments of hard manganese steel or cast basalt. Where wet coal is

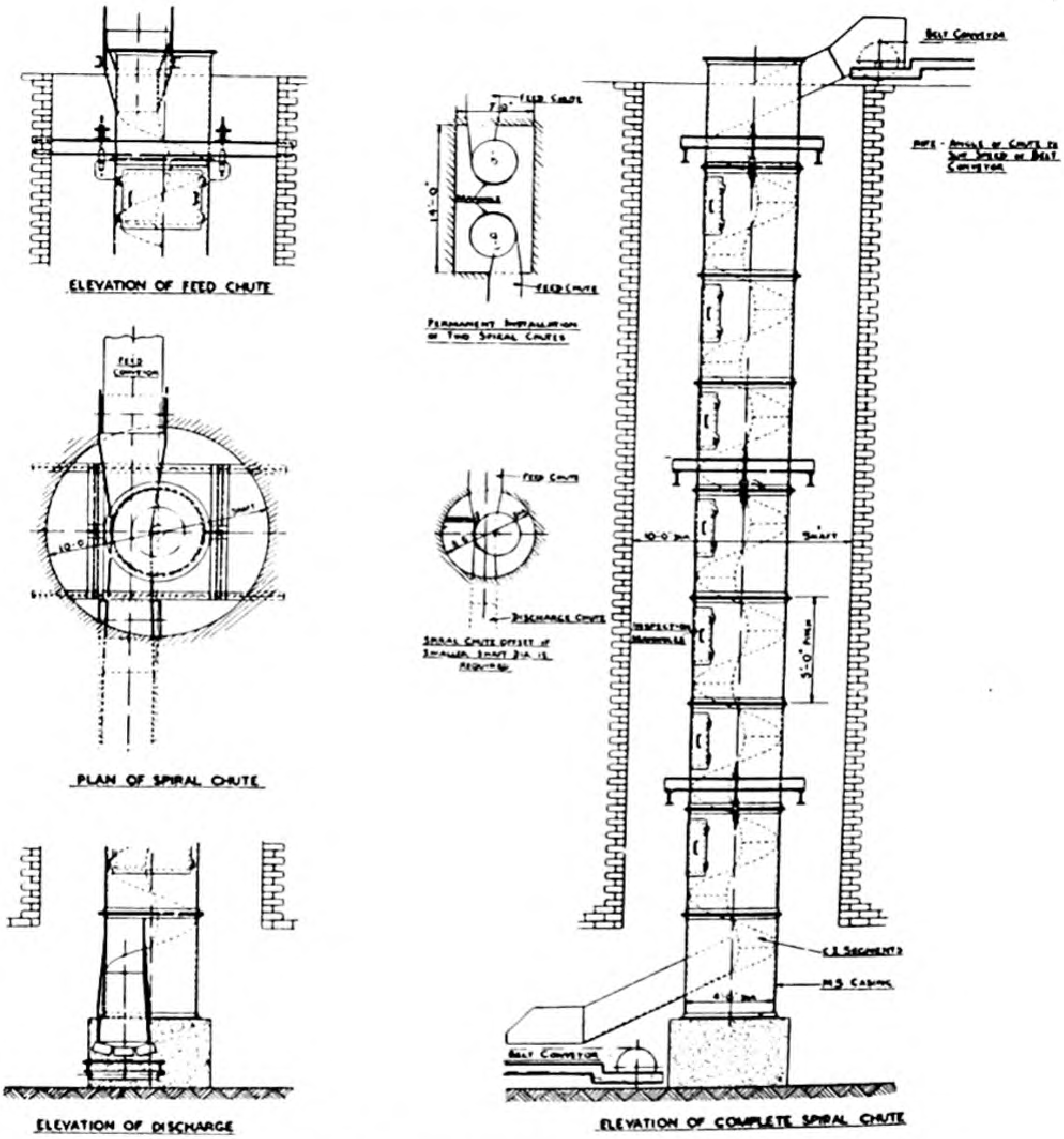


FIG. 274.—Typical arrangement of Qualter Hall spiral chute.

being conveyed or the staple shaft itself is wet, a cast basalt lining is preferable, as hard manganese steel will rust, even if the chute is out of operation for only one day, resulting in an early deterioration of the inserts, blockages and an increased production of fines. The life of the spirals is increased by using such resistant linings to give a

throughput of from 1·5 to 2 million tons, and it is the general practice to use them in the Ruhr where the conditions warrant it.

The inserts, or linings, are attached to the spiral sections by countersunk screws, while the segments may be dovetailed together. The inserts are usually fixed after the spiral chute has been installed in the shaft. The actual chute installation takes about three or four days for a chute 100 yards long. The fastening of the spiral segments can be carried out, at a rate of 50 segments per manshift, assuming that the pitch of the spiral is about 29 inches. The inlet, or feed chute, is usually fitted in place of a normal spiral chute section and is

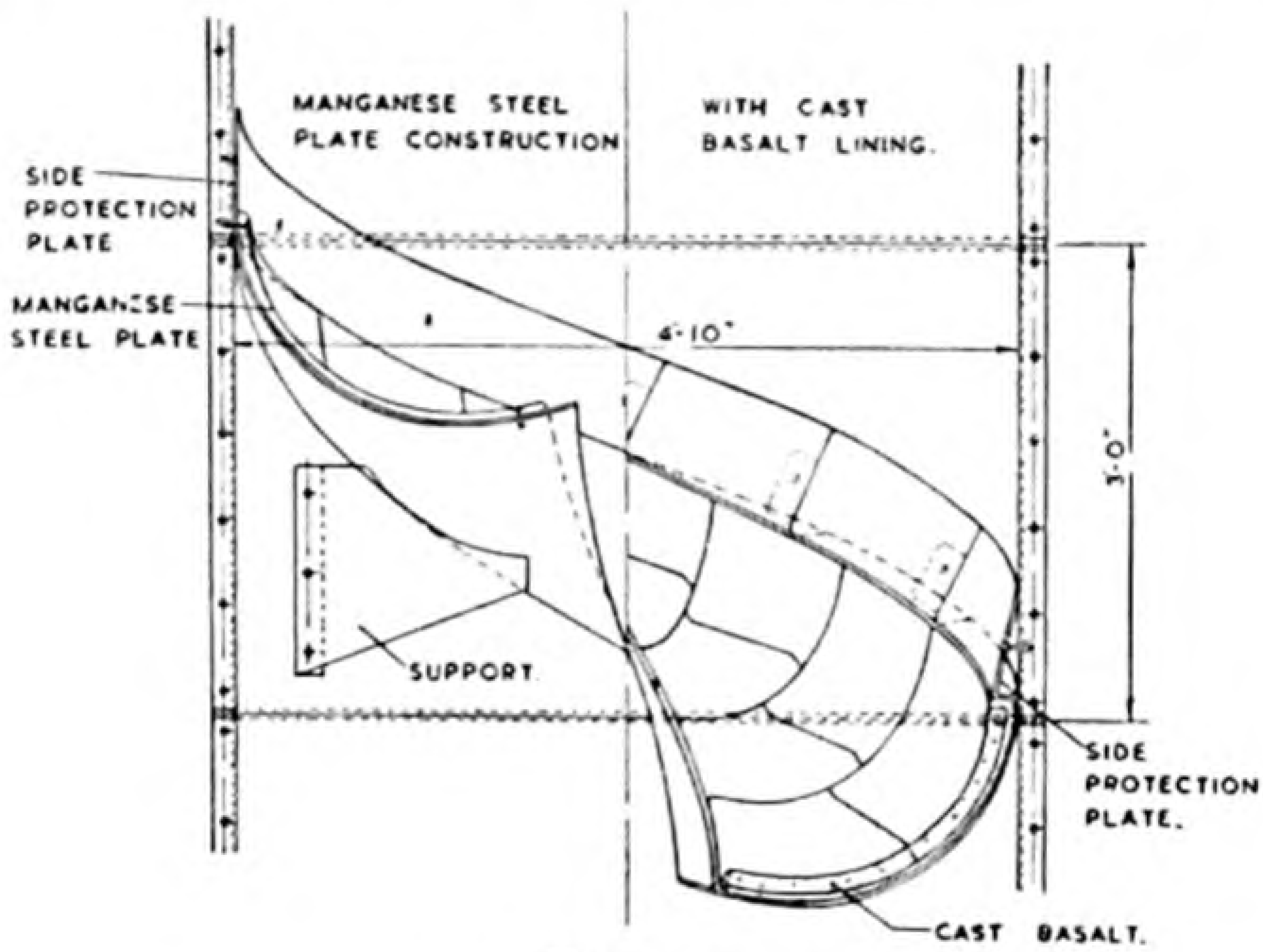


FIG. 275

designed to deliver the coal in the direction of the upper spiral section, as shown in Fig. 276. Where the spiral chute is fed from two sides, the feed chutes must be staggered, as in Fig. 277, while additional intermediate feed chutes can be inserted at any particular level. The discharge chute, shown in Figs. 274 and 278, is also fitted in place of the ordinary chute section. There are a number of designs, the usual one incorporating a totally enclosed chute tapered to the mouth and having an inclination of about 30 degrees. The control of the discharge from the mouth of the chute can be carried out by cut-off doors operated mechanically by compressed air or by a hand-operated balanced guillotine. In some cases the delivery chute serves a feeder of either the compressed-air-operated reciprocating pattern or the Sherwen type. In the case of the reciprocating type, a cut-off door is not absolutely necessary, since on stopping the

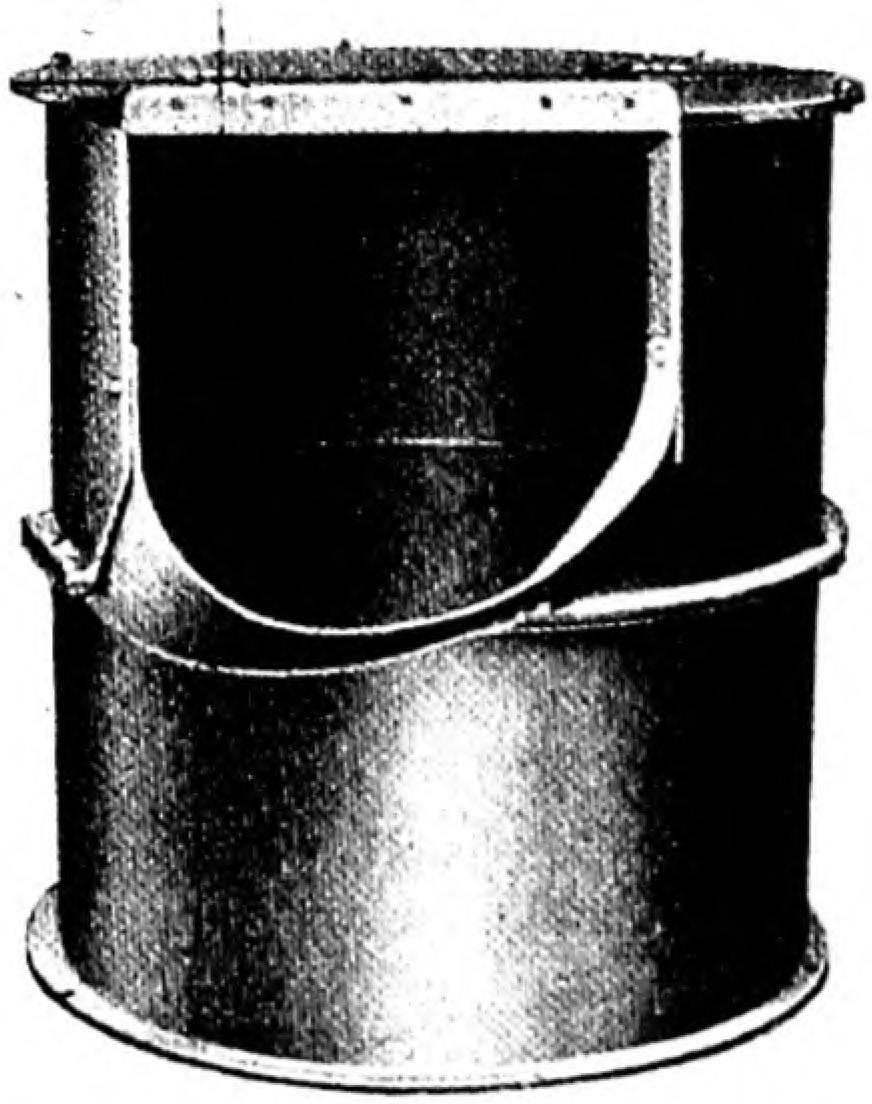


FIG. 276

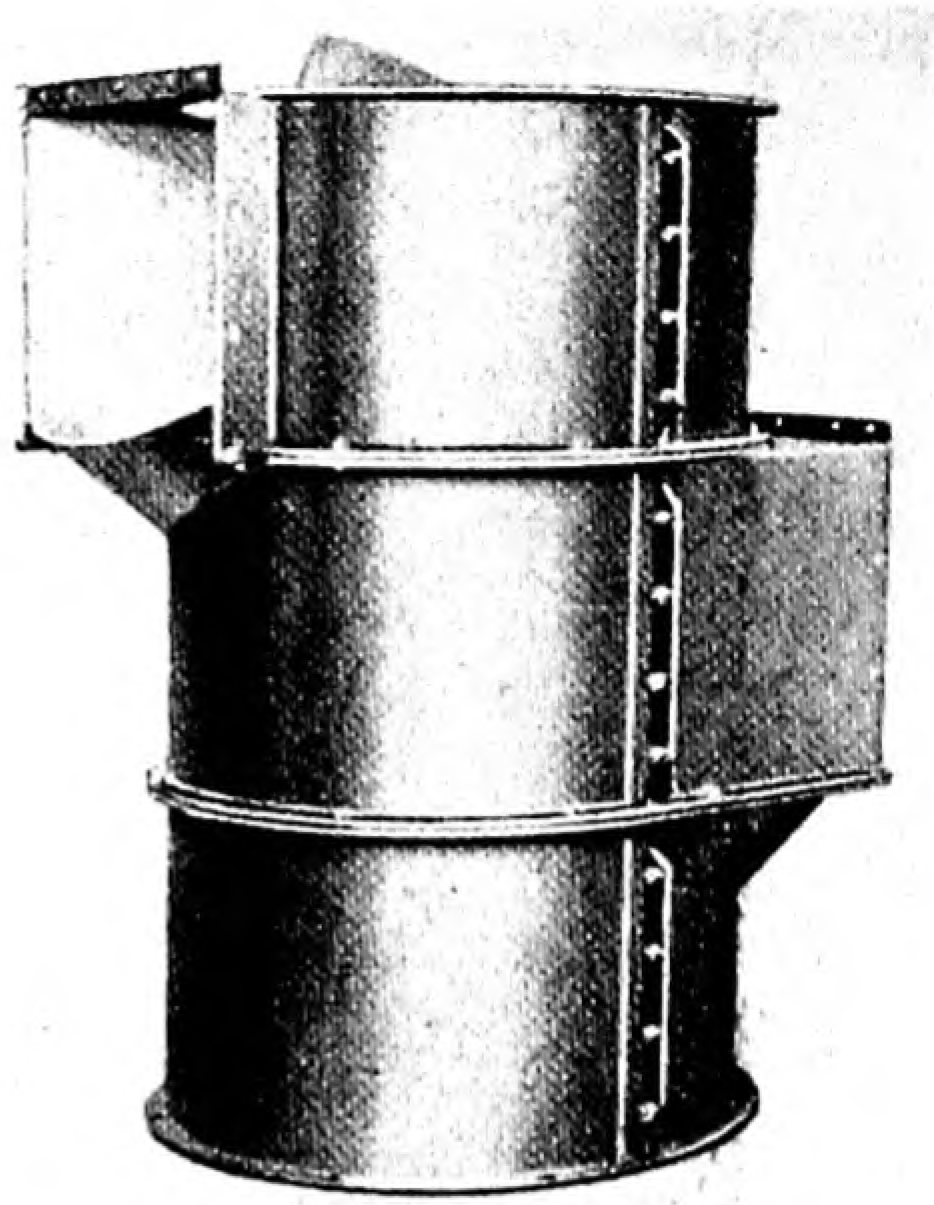


FIG. 277

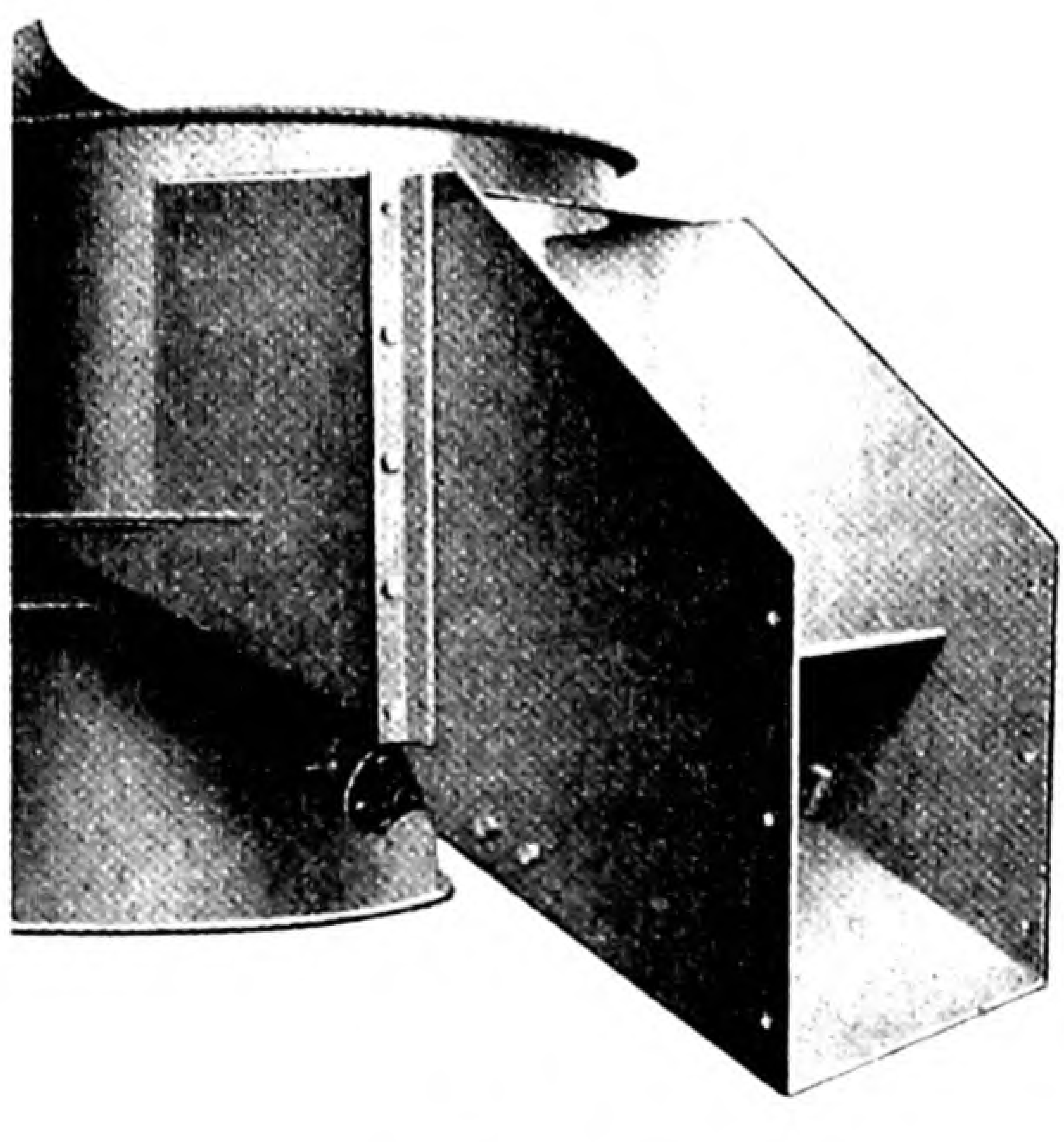


FIG. 278

feeder the coal accumulates on the deck and prevents further delivery. The coal may be delivered direct into mine cars or on to a belt or shaker conveyor. Several forms of spiral-chute delivery installations are illustrated in Figs. 279, 280 and 281.

The capacities of British and Continental spiral chutes are given in the previous table. The choice of diameter of the chute required depends more on the size-range of the coal to be handled than on the capacity of the haulage system. Reference should be made to the table below, in which capacities are given in relation to diameter and maximum coal size. The greater the proportion of lump coal, say above 4 inches, the greater the possibility of having a blockage in the chute, and for this reason the 4-foot 1-inch (1,250 mm.) diameter chute is the commonest size in use in the Ruhr. It is known that pieces 3 feet by 3 feet by 1 foot 8 inches can be passed through this size of chute.

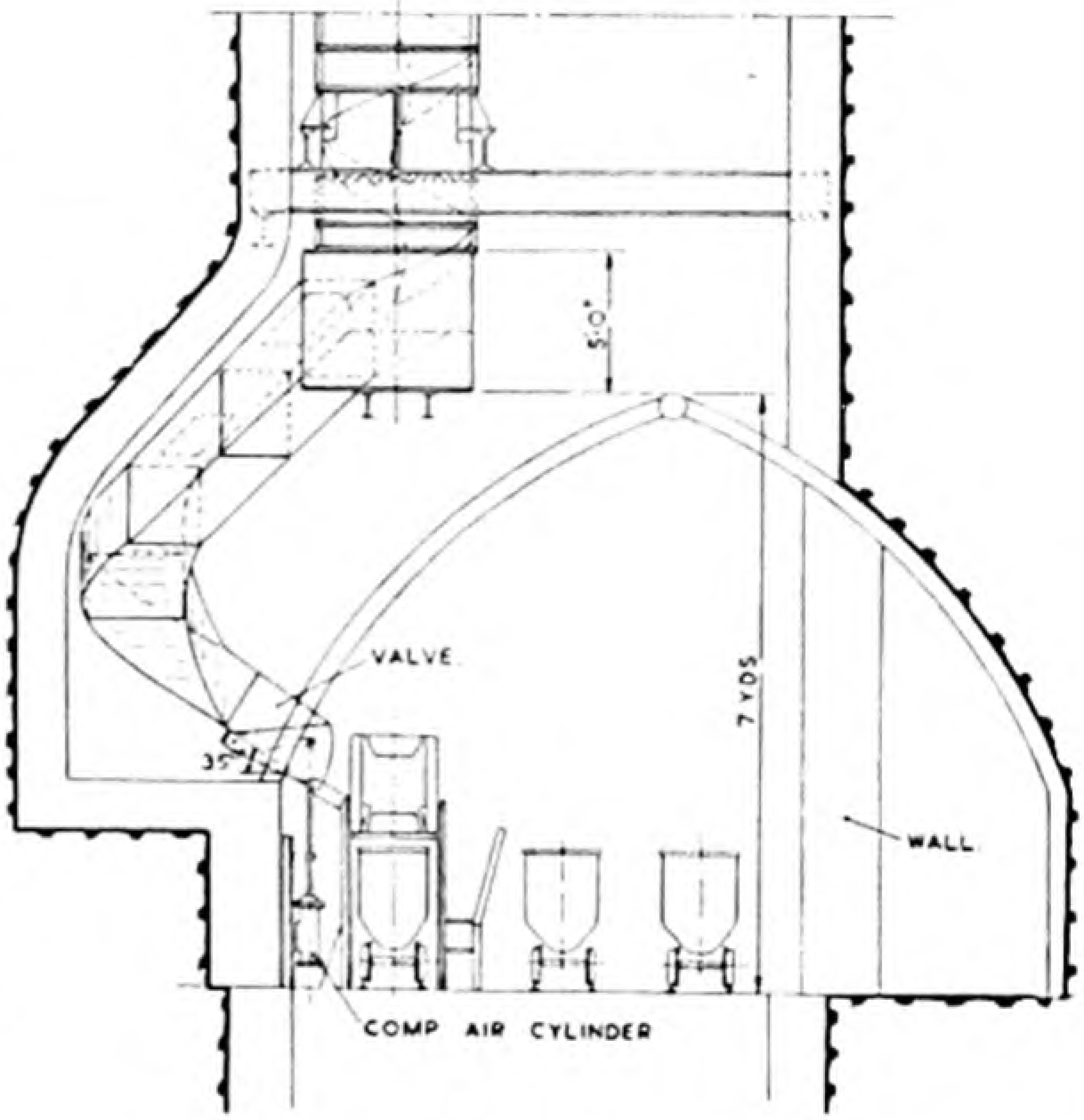


Fig. 279

The bunkering capacity of spiral chutes can be taken advantage of in minimising stoppages for minor haulage delays. The capacities per yard run for the three main sizes used in the Ruhr are as follows:

Diameter	3 *ft.* 5$\frac{5}{16}$ *in.* (1,050 *mm.*)	4 *ft.* 1 *in.* (1,250 *mm.*)	4 *ft.* 9 *in.* (1,450 *mm.*)
Capacity per yd. run . . ton	0·6	0·8	1·0

A main disadvantage of conveying by spiral chute is the disintegration and degradation of the coal so that the fines are increased

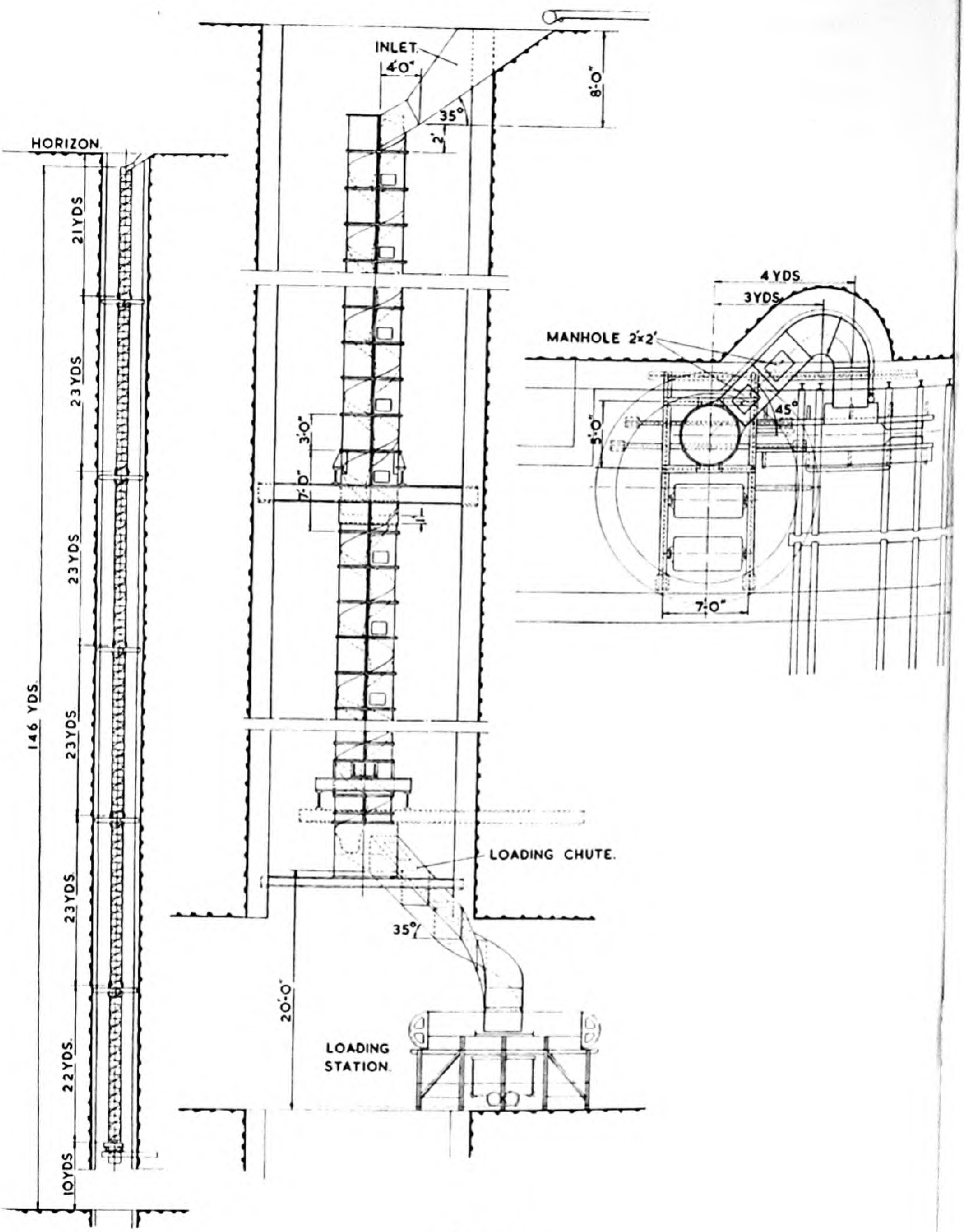

INLET.
4'0"
8'0"
35°
2'
HORIZON
21YDS
23YDS
23YDS
23YDS
23YDS
22YDS.
10YDS
146 YDS.
3'0"
7'0"
1'
4 YDS
3YDS
MANHOLE 2'x2'
45°
5'0"
7'0"
LOADING CHUTE.
35°
20'0"
LOADING
STATION.

Fig. 280

at the expense of reducing the proportion of larger sizes. The increase in fines may vary within the wide limits of 2 to 20 per cent., depending upon the nature of the coal and the efficiency of the spiral-chute design in preventing breakage. The hardness and brittleness of the coal, the cleavage and the presence of dirt influence the

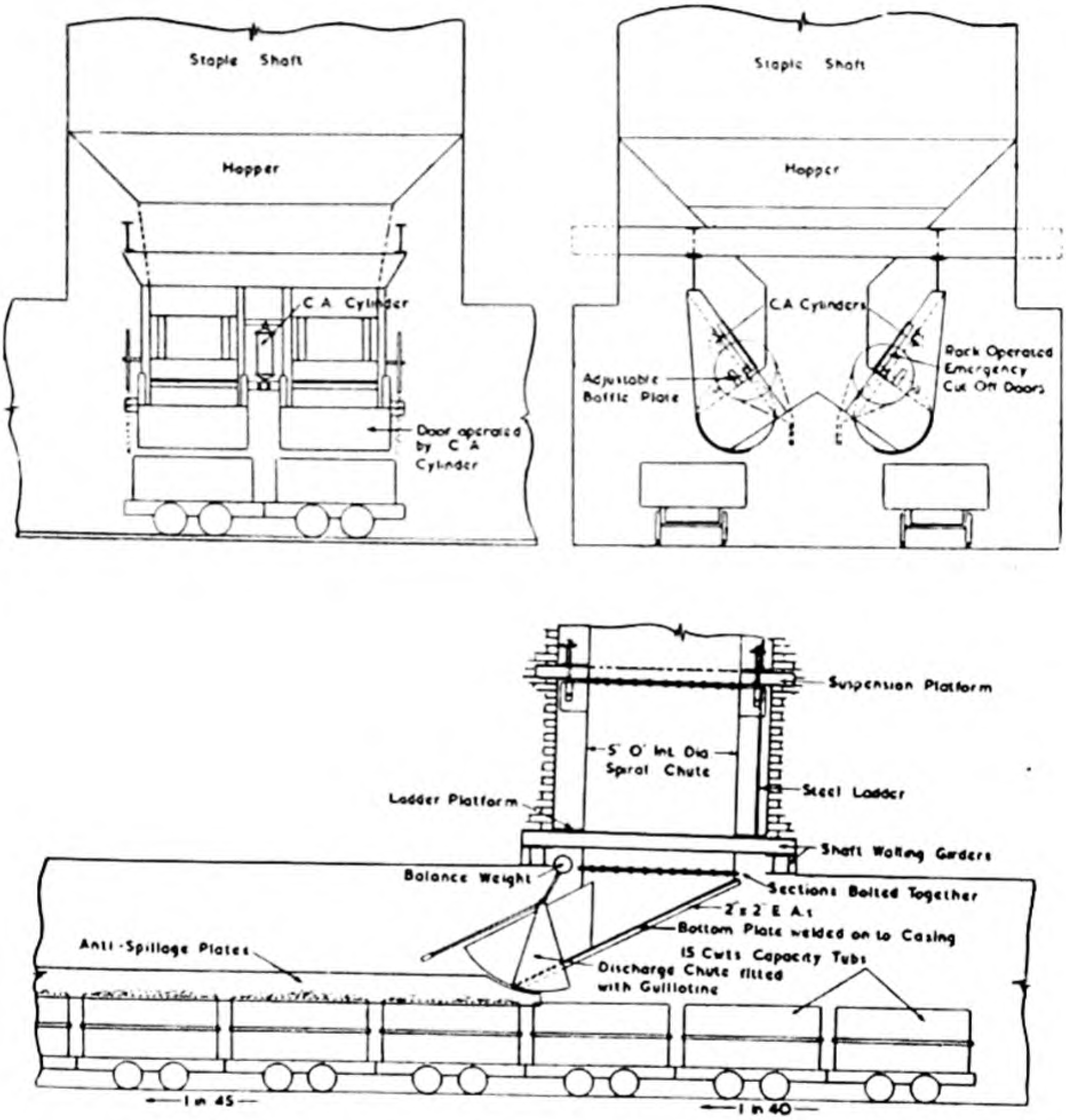


FIG. 281

extent of the final size delivered. Tests conducted by Hartung (*Glückauf*, *3rd February*, 1951) on a 3 feet 6 inches diameter chute showed that the unmixing which had the greatest effect on degradation happened during the first 12 yards and that from this point onwards, it was caused by friction on the chute plates. Since dirt has a high coefficient of friction, it slows down the movement, and

Hartung shows that there is less degradation with dirt present after the first 12 yards. A hard or strong coal will suffer much less degradation than a medium, soft or brittle coal. The effect of the presence of dirt or pit waste will be greater, the larger the stones among the coal. The influence of such material may be compared with the use of steel balls in a ball mill. The waste material itself will be reduced to nut sizes rather than to fines. When conveying coal containing a low percentage of waste or dirt, the production of fines will be approximately proportional to the height of the chute. The diameter of the spiral chute also has an influence on fines production. Experience in the Ruhr has shown that disintegration is less in a heavily charged chute than when it is operated at less than its full capacity. Since the larger diameter of chute is normally chosen to minimise the possibility of blocking, the result is that, in the main, the chutes are run well under capacity and more disintegration results than would be the case if chutes of smaller diameters were used.

The condition of the sliding surface of the chute has already been discussed in relation to corrosion, and it is true to say that the smoother and less uneven the surface, the more even the wear and the longer the life of the spiral section.

(*b*) *The vertical conveyor.* The vertical conveyor, like the spiral chute, provides a continuous method of transport of material in staple shafts. As shown in Fig. 282, it is designed to convey from the upper to the lower level and is comprised of a tube of rectangular cross-section, called the 'lock', within which the coal is carried down in compartments. The compartments are formed between the walls of the casing, or lock, and the straight or slightly curved retaining flaps. The retaining flaps are attached about 3 feet apart to two parallel Reynolds'-type chains driven at about 1·5 feet per second from appropriate sprockets on the drive shaft of the engine at the upper level. The retaining flaps are free to move vertically on the chain, and lean obliquely against the front side of the casing during the downward movement and fall against the chain when going upward on the empty side. The casing and retaining flaps are also designed in trapezoidal section, with the short side to the outside so that the same seal is effected with the flap and casing side. The rectangular lock, or casing, is constructed of mild-steel plate, the side walls being $\frac{3}{16}$ inch thick and the front side $\frac{1}{4}$ inch thick. The

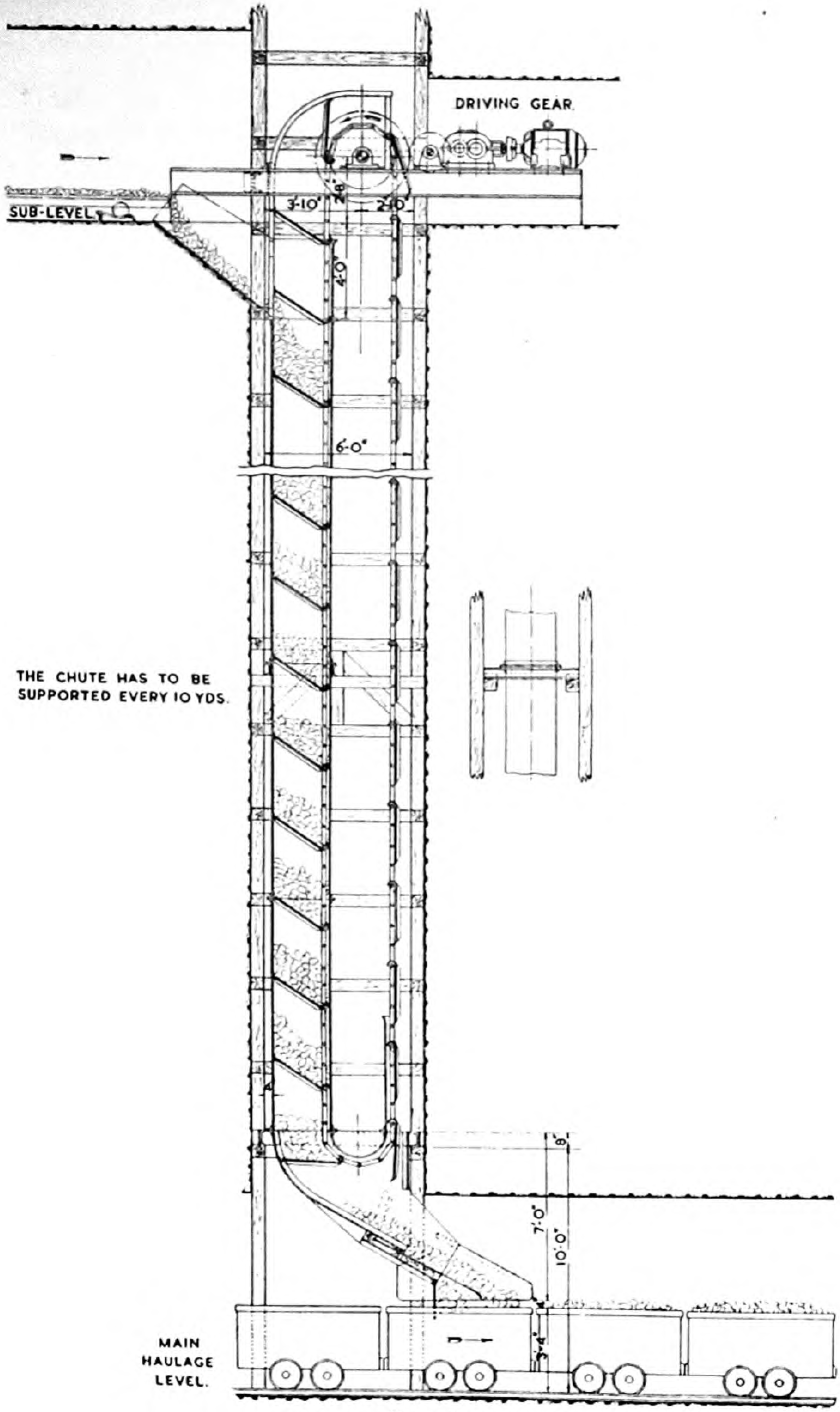

DRIVING GEAR.
SUB-LEVEL.
3'-10"
2'-8"
2'-10"
4'-0"
6'-0"
THE CHUTE HAS TO BE SUPPORTED EVERY 10 YDS.
8"
7'-0"
10'-0"
3'-4"
MAIN HAULAGE LEVEL.

FIG. 282

casing has a cross-section of 2 feet by 2 feet 4 inches, and is built up in 10-foot sections bolted together so that the lock can be opened up for inspection. The chain links are 10 inches long, and for installations of 80, 100 and 120 yards deep, the chain used has a breaking strength of 35, 42 and 55 tons respectively. The inlet is a simple feed chute, while the discharge bunker is usually designed for individual circumstances and is made of steel plate from $\frac{1}{4}$ to $\frac{1}{2}$ inch thick; one design is illustrated in Fig. 282.

The operation of the conveyor requires a continuous and regular braking effort. It is obvious, therefore, that ordinary friction braking should not be adopted, but the sprocket shaft should be coupled to a squirrel-cage motor so that electricity is generated when braking is necessary and this is fed back into the line. Where the drive is by a compressed-air engine, it is necessary to use an engine which, after reaching the maximum speed required to operate the conveyor, operates as a compressor and produces a braking effect. The application of the braking effort applied to the system, either by the operation of the motor as a generator or the compressed-air engine as a compressor, must be carried out smoothly, otherwise the change may result in a chain breakage, and for this reason an electric-motor drive is more suitable. The braking capacity of the motor required depends upon the depth, but in all cases a positive brake for stoppage is necessary. For a depth of 55 yards with a loading capacity of between 150 and 200 tons per hour, the brake horse-power required will be between 16 and 21, while at a depth of 110 yards with the same load, the brake horse-power necessary would be between 36 and 48. The special advantage of the vertical conveyor over the spiral chute is that, due to the controlled delivery, much less breakage occurs, with a consequent reduction in dust formation. The wear is reasonably small, so that throughputs of 1·5 million tons can be expected without heavy maintenance or replacement costs. The conveying distance is limited by the breaking strength of the chains. Generally the limit is taken as 120 yards, although there are cases where the conveyor has operated satisfactorily up to depths of 165 yards. With greater depths of staple shaft it is recommended that two vertical conveyors in tandem be installed, as shown in Fig. 283. The capacity of the conveyor is dependent on the speed and cross-sectional area of the casing used. The construction described will deal with 250 tons per hour.

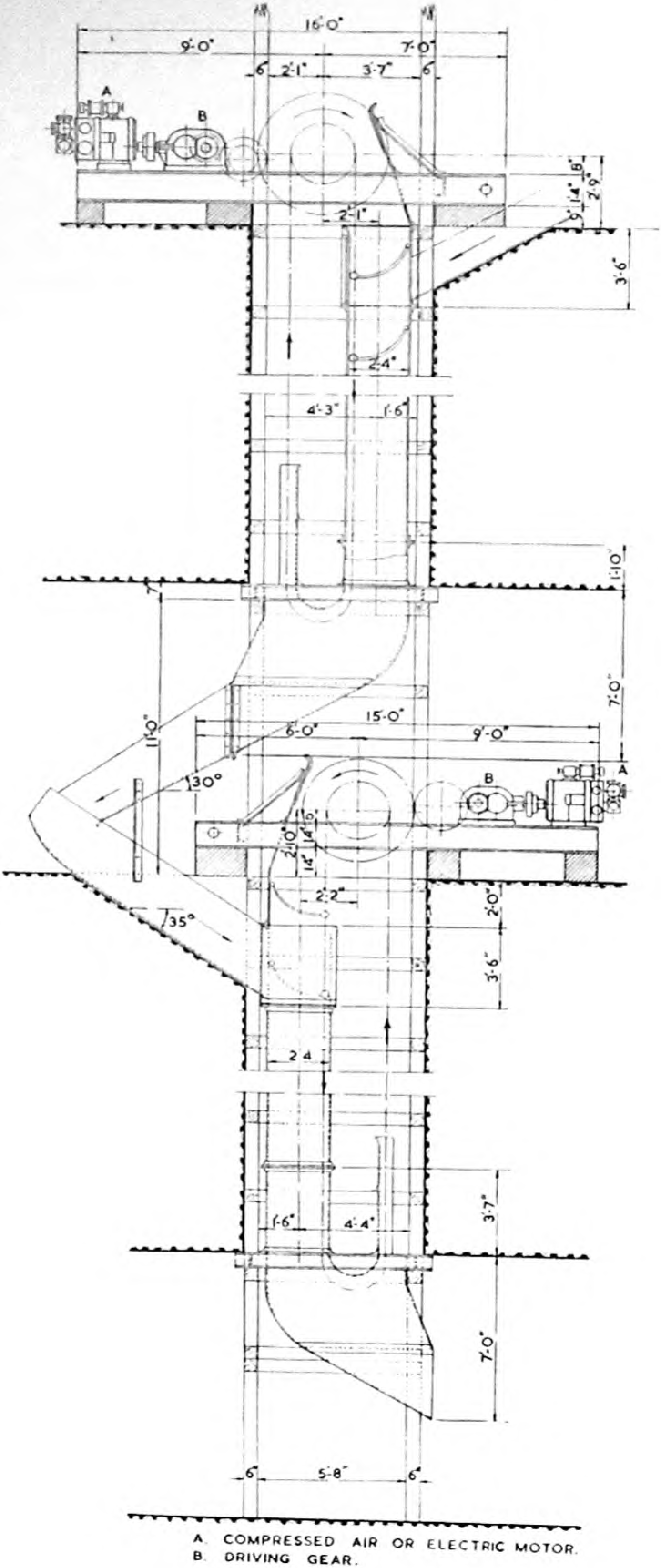

16'-0"
9'-0"
7'-0"
6"
2'-1"
3'-7"
A
B
8"
1'4"
2'9"
9"
2'-1"
3'-6"
2'-4"
4'-3"
1'-6"
1'-10"
7'-0"
11'-0"
15'-0"
6'-0"
9'-0"
30°
35°
2'-10"
1'4"
5"
2'-2"
2'-0"
3'-6"
2'4
3'-7"
1'-6"
4'-4"
5'-8"
A. COMPRESSED AIR OR ELECTRIC MOTOR.
B. DRIVING GEAR.

FIG. 283

Section 10. *Man-riding in Staple Shafts*

Most staple shafts are equipped for man-riding in order to provide easy and speedy access to the working levels, to maintain the available man-working time at the maximum possible and to assist supervision. In view of the heavy transport demand on staple-shaft hoisting systems, strict control of the hoisting operations and the adoption of the necessary safety measures are absolutely necessary. Where men are also being wound, the normal legal precautions must be observed, even if the maximum hoisting speed is restricted to from 6 to 12 feet per second, below which speed automatic over-wind prevention is not demanded. Special regulations for staple-shaft man-riding are not yet drafted, so that the general winding regulations are applicable as far as they can be practically applied.

It is recommended that a sump should be provided at least 6 feet below the cage bottom at the hanging-on level, within which the distance between the gliding beams, or guides, should be gradually reduced. The sump should be kept free of water in case the cage is at any time dropped too far. A similar precaution should be taken at the banking-out level, and in this upper extension of the shaft the guides should be brought closer together and reinforced; keps should not be used. Ordinarily, detaching gear is required for each cage where it is being used for raising or lowering men. If the maximum speed is limited to less than 12 feet per second, the regulations require the provision of a reliable depth indicator and a self-acting brake adequate to hold the loaded cage in mid-shaft. Should the winder be capable of a speed exceeding 12 feet per second, even if the man-hoisting speed is restricted to less than this velocity, over-wind-prevention gear is required to limit the landing speed to less than 5 feet per second.

In addition to these legal requirements, it is desirable to incorporate a reliable ammeter and voltmeter. The diameter of the winding pulleys used should be at least forty times the rope diameter. Each winding rope has to be maintained according to the regulations governing their use, as stated in Parts 1, 2 and 3 of the General Regulations, 1937. The cages used should have a strong protective roof and have steel-sheet sides. Gates must be provided on the open sides and secured against accidental opening during winding.

Section 11. *The Layout of Staple-shaft Bottoms and Sidings for Cage Winding*

In laying out the staple-shaft bottom and sidings, the location of the staple shaft with respect to the position of the main road of the winding level, and the strike of the seam, have a decisive influence. If the staple shafts are situated in or near a cross-cut, the axis of the shaft may be either parallel or at right angles to the direction of the cross-cut. The shaft axis is referred to the direction in which the tubs are being pushed into the cage. If the shaft axis is parallel to the line of the cross-cut, the tubs may be run in and out of the cage in the cross-cut direction, without having to turn through a right-angled curve. At the upper landings, such convenience in the decking of the cages is possible only when a landing is laid out in a sub-level cross-measure drift. This condition occurs only in steep measures, while in flat-seam formations the landings are in the immediate vicinity of the seams. The gate roads driven in the seam begin at the staple shaft, and since they are at right angles to the level cross-measure drift, the tubs which run between the staple shaft and the gate roads must negotiate a right-angled curve. In such conditions a special landing layout is required to provide for a single change of tubs and to enable quick straight-through decking to be carried out. The disadvantages of a special layout may be avoided by locating the staple-shaft axis at right angles to the direction of the level cross-cut.

If the staple-shaft bottom is on a main lateral road at the winding level, the gate roads are driven in the same direction as the main road of the winding level. In this case the shaft axis is parallel to the level, and straight-through decking is possible without any difficulty either at the hanging-on level or in the sidings. The layout of a staple-shaft bottom in which the shaft axis is parallel with the main cross-measure drift is shown in Fig. 284. It will be seen that the shaft-winding compartments are within the drift width and the necessary widening of the road to take sidings track is reduced to a minimum. The track in front of and behind the shaft must provide sufficient accommodation to take at least one empty and one full train and, if possible, the standage should be more to allow for delays in tub circulation. The full benefit of ample siding accommodation cannot always be secured, due to the proximity of other staple shafts,

but, even in this event, the siding accommodation provided for full and empty tubs should not be so restricted as to affect the staple-shaft drawing capacity. In such cases shorter trains should be operated more frequently to and from the shaft, although longer and less frequent trains would usually be sufficient. In order to make full use of the locomotives, the normal number of tubs required should be allocated equally between them, and it may be necessary, if the siding room is restricted in capacity, that one locomotive serves two neighbouring staple shafts. A disadvantage of such an organisation is the necessity to make the locomotives do more shunting operations, with a consequent increase in the waiting time.

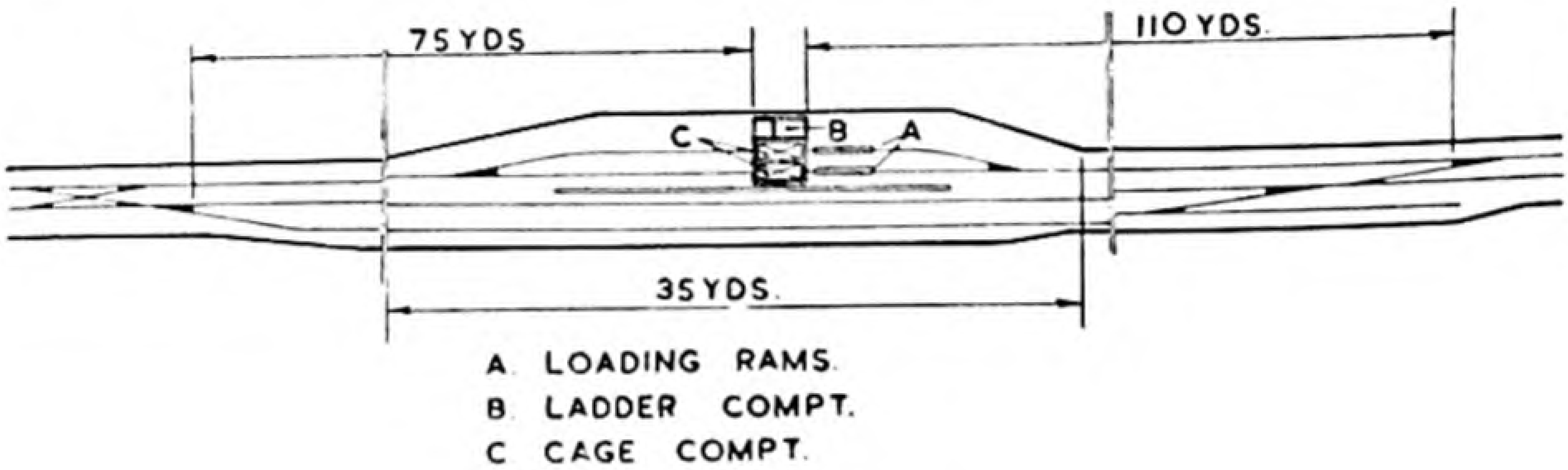


FIG. 284

It is also important to consider whether the existing roadway track should be used as siding accommodation for full and empty tubs or whether additional track should be laid. If only single track is employed, as may happen where large mine cars are being used, an additional track is imperative. Where, however, double track is laid, it would be possible to use one track at the side of the shaft for siding accommodation. This can be done if the other track is sufficient to deal with traffic to and from other staple shafts or where there is no such traffic at all. The advantage of such an arrangement is obvious, since any widening of the main road is only required near to the shaft where the third track is necessary for the second winding compartment. In order to provide additional roof support where the road is widened near the shaft, a central wall, one and a half or two bricks wide, is usually built, as shown in Fig. 284. Where the traffic past the staple shaft is heavy, the addition of a third track for standage cannot be avoided.

Another layout with the shaft axis still in line with the main road is shown in Fig. 285, in which the shaft is located about 10 yards

off the main road in a parallel bypass. The disadvantage of this layout lies in the additional drivage required, and for this reason it is not frequently carried out. It may be necessary, however, to lay out the shaft and sidings in this manner where the main road in the upper level is not immediately above that in the lower level. A feature of this layout is that the shaft bottom is off the main intake, which is an advantage if the shaft bottom is a main loading-point, especially where a spiral chute is installed. Where the staple shaft is situated at the end of a main haulage level, the layout shown in Fig. 286 may be adopted. If the shaft axis direction is at right angles to the main haulage level, the layout shown in Fig. 287 may be used. In this layout the shaft is located in a short cross-cut about 6 to 8 yards from the main level. The difficulty in securing easy tub circulation is overcome by using an acute-angle return-point behind the

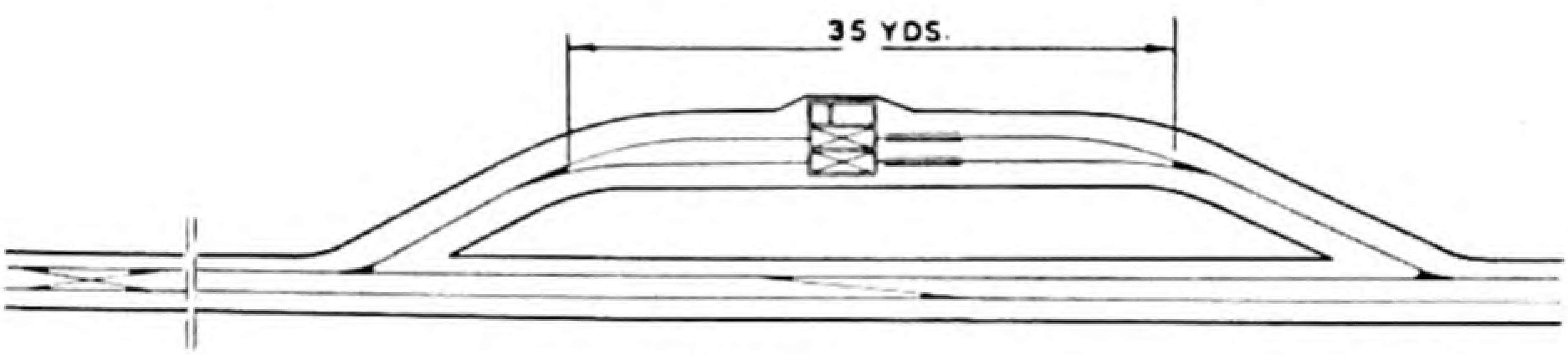


FIG. 285

shaft. The full tubs from the cage are pushed out to the back of the shaft, the track rising to the back of the cross-cut, then run back from a buffer past the automatic spring switch and pass back to the empty-tub siding. The additional track behind the shaft may be laid as in Fig. 287, but if drivage of the cross-cut is restricted to the width necessary for the shaft itself, the layout can be carried out as in Fig. 288, in which the full gathering track runs under the ladder compartment, and this gives a very compact and favourable system.

Another possible layout is illustrated in Fig. 289, which combines the two previous schemes, incorporating the diagonal gathering track for full tubs with the cross-cut driven at shaft width. This system is obviously more expensive to carry out, since the diagonal road is additional.

(*a*) *Sub-level shaft insets.* At the intermediate landings where tub drawing has to be carried out, similar arrangements must be made to secure quick tub circulation and ensure the full drawing capacity of the staple shaft. If the shaft axis is at right angles to the gate road,

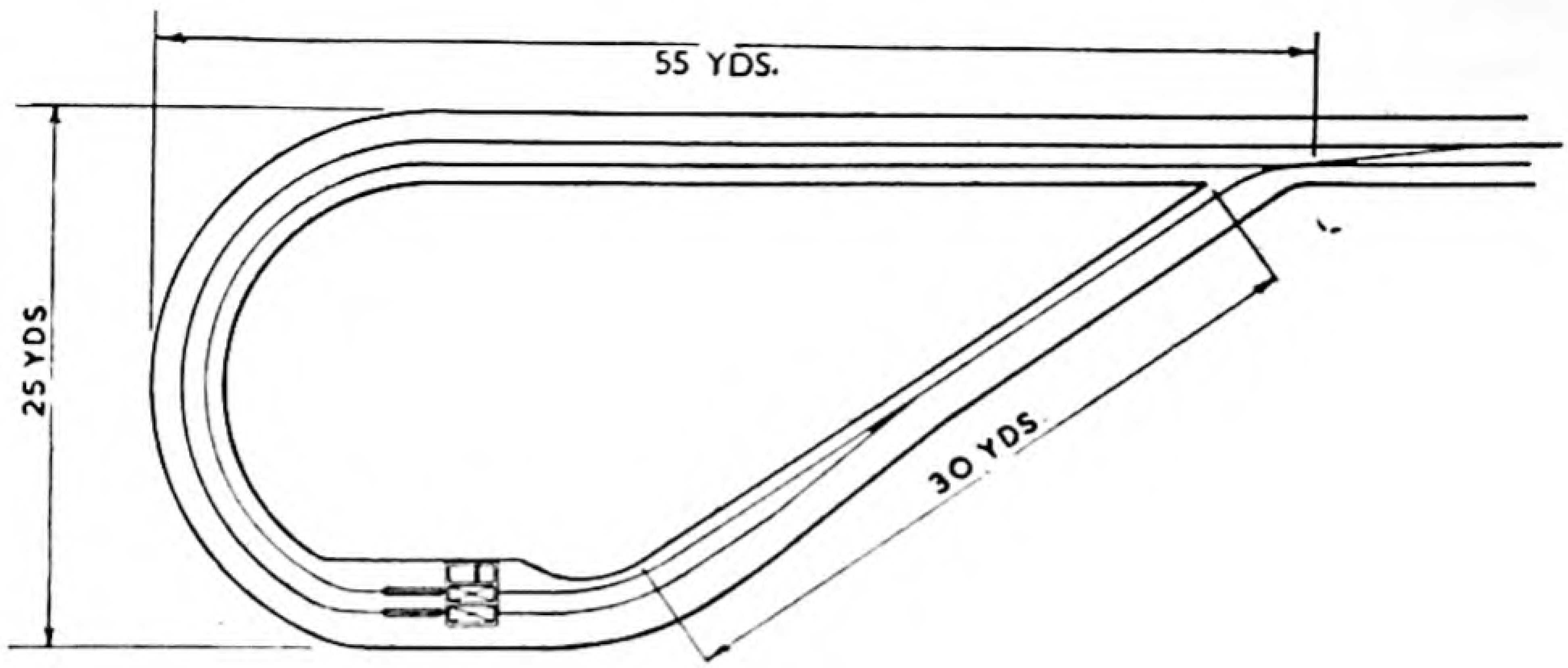


FIG. 286

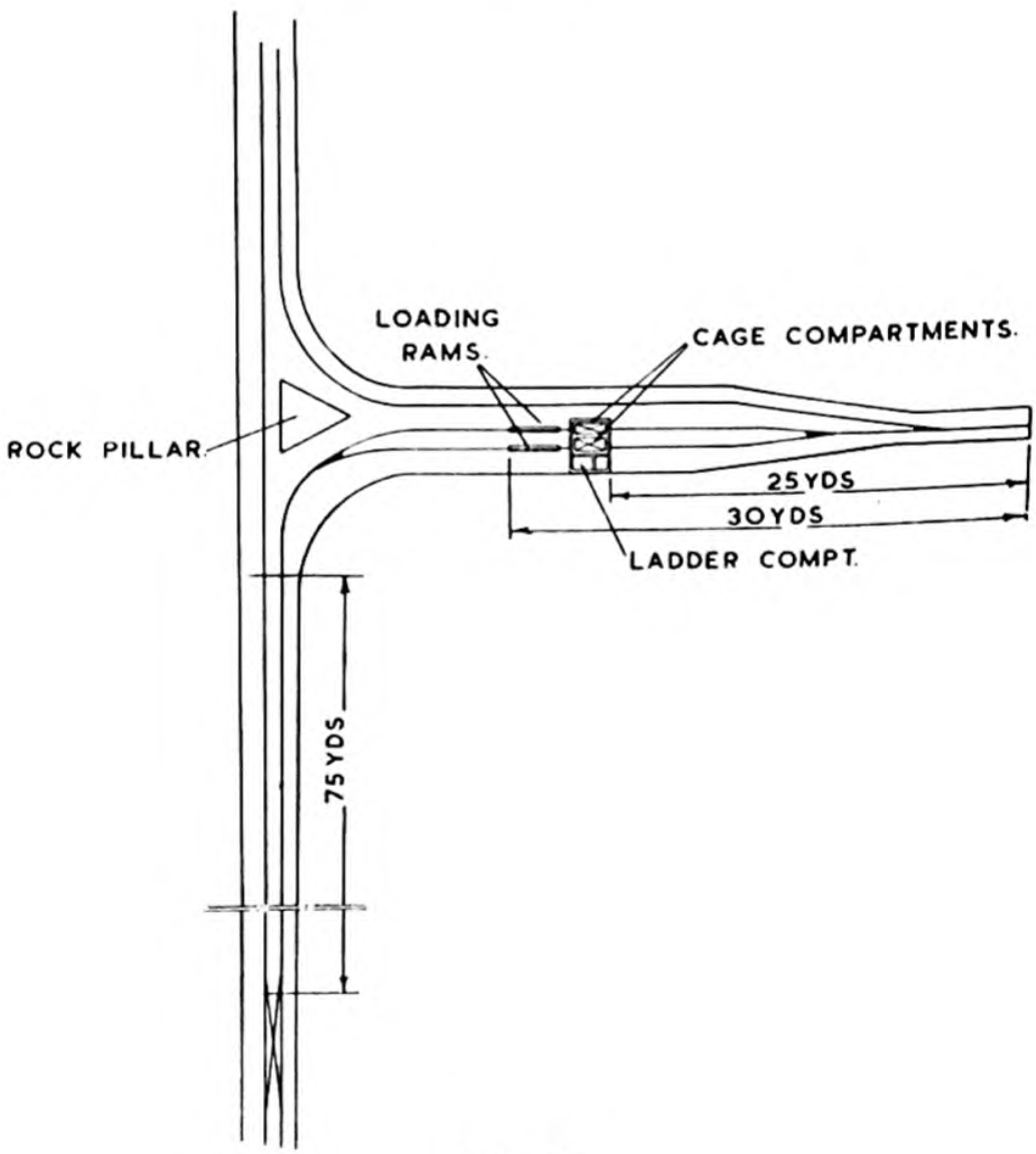


FIG. 287

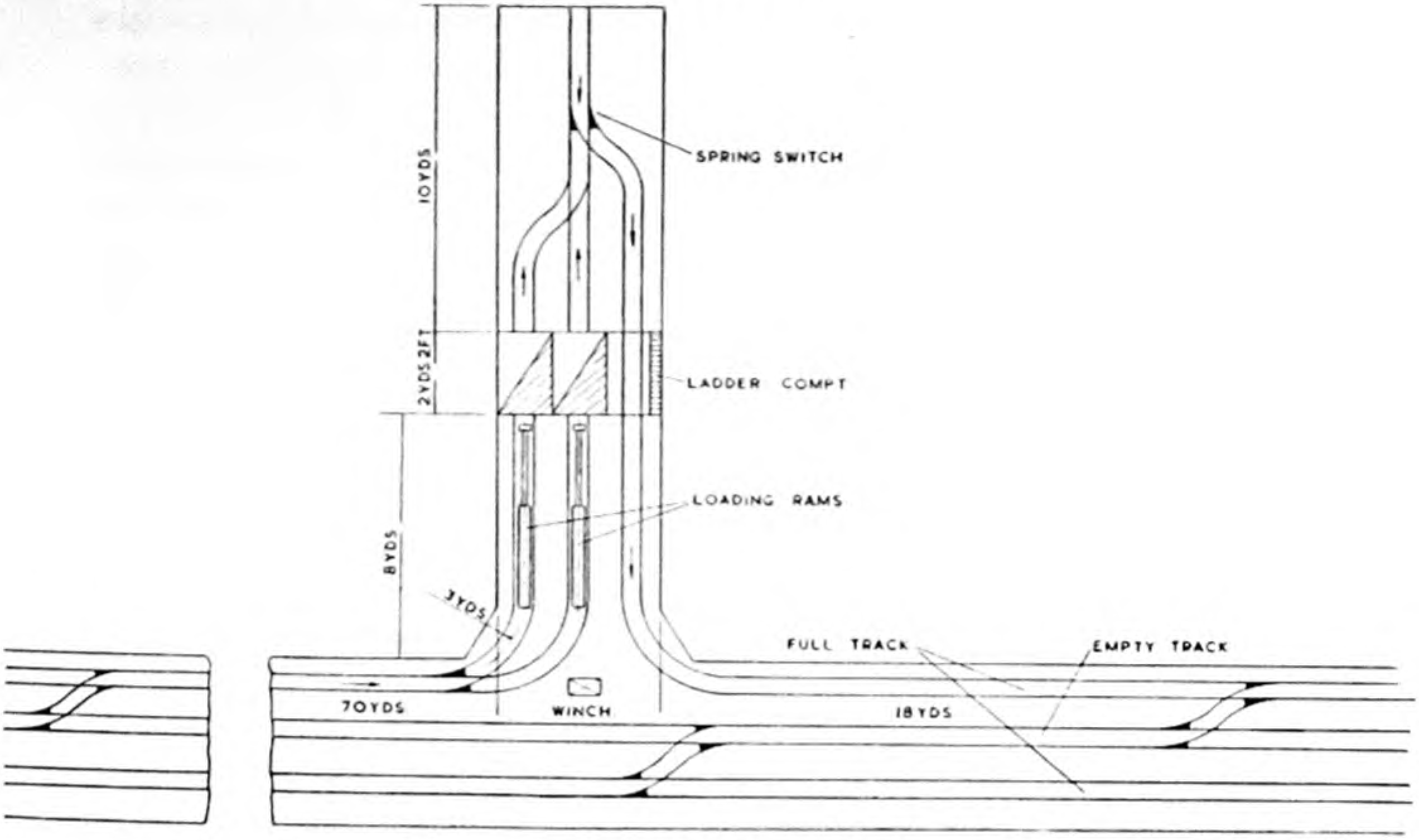


FIG. 288

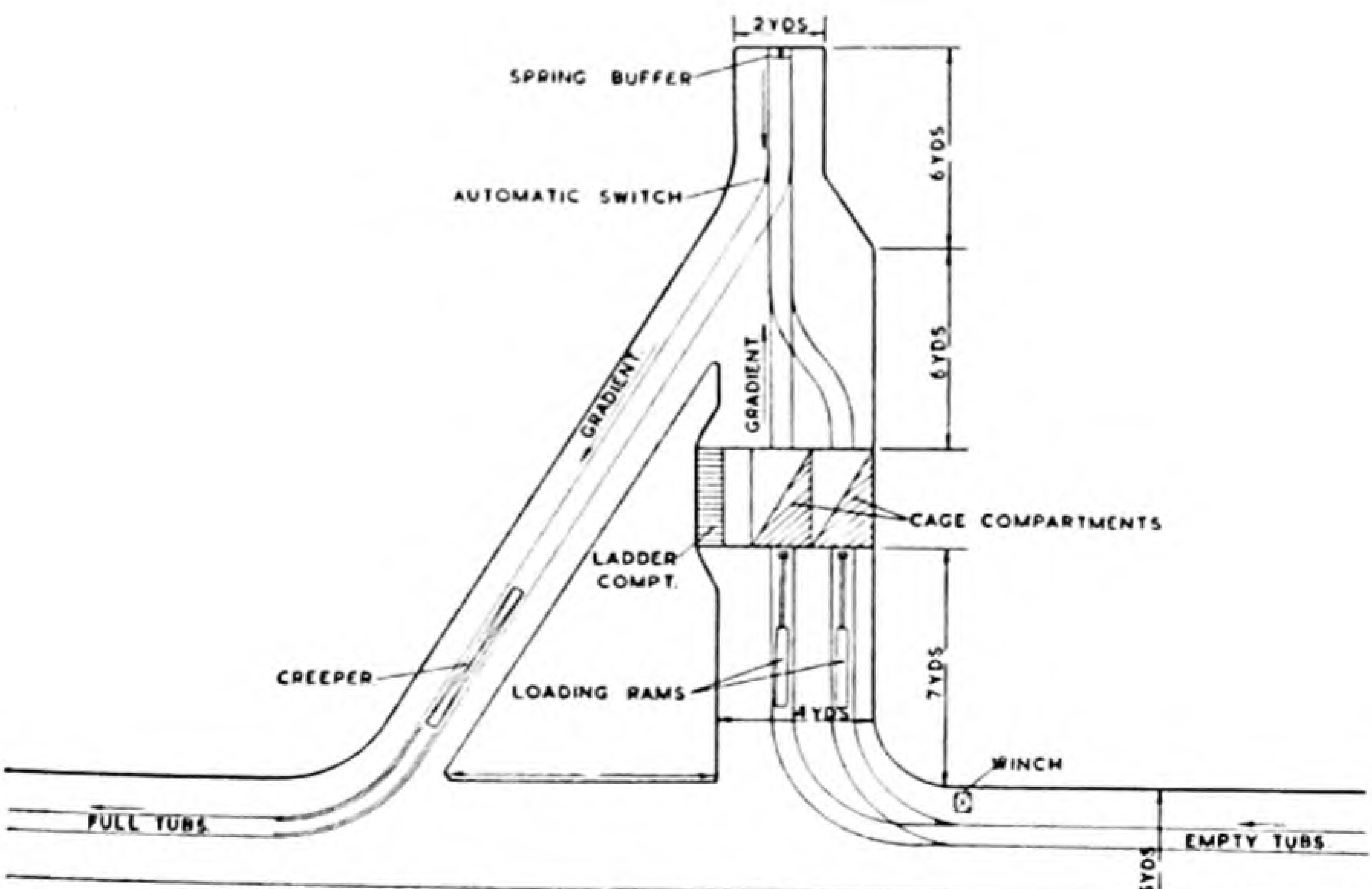


FIG. 289

straight-through circulation can only be achieved by using bypasses as shown in Fig. 290. Where the shaft axis corresponds to the gate-road direction, bypasses are not necessary and it is sufficient, in these circumstances, to arrange that the tubs leaving the cage pass beside the shaft and are fed to their respective gate road. If mechanical decking devices are used and two sides are worked from two gate roads, the layout must incorporate arrangements for decking the full tubs always from the same side. The same problem arises, as in main-shaft winding, of a supply of full tubs from both sides of the shaft. Two different layouts to overcome the problem are shown in Figs. 291 and 292.

In the example shown in Fig. 291, small-radius curves are utilised on each side of the shaft, around which the tubs are drawn by a

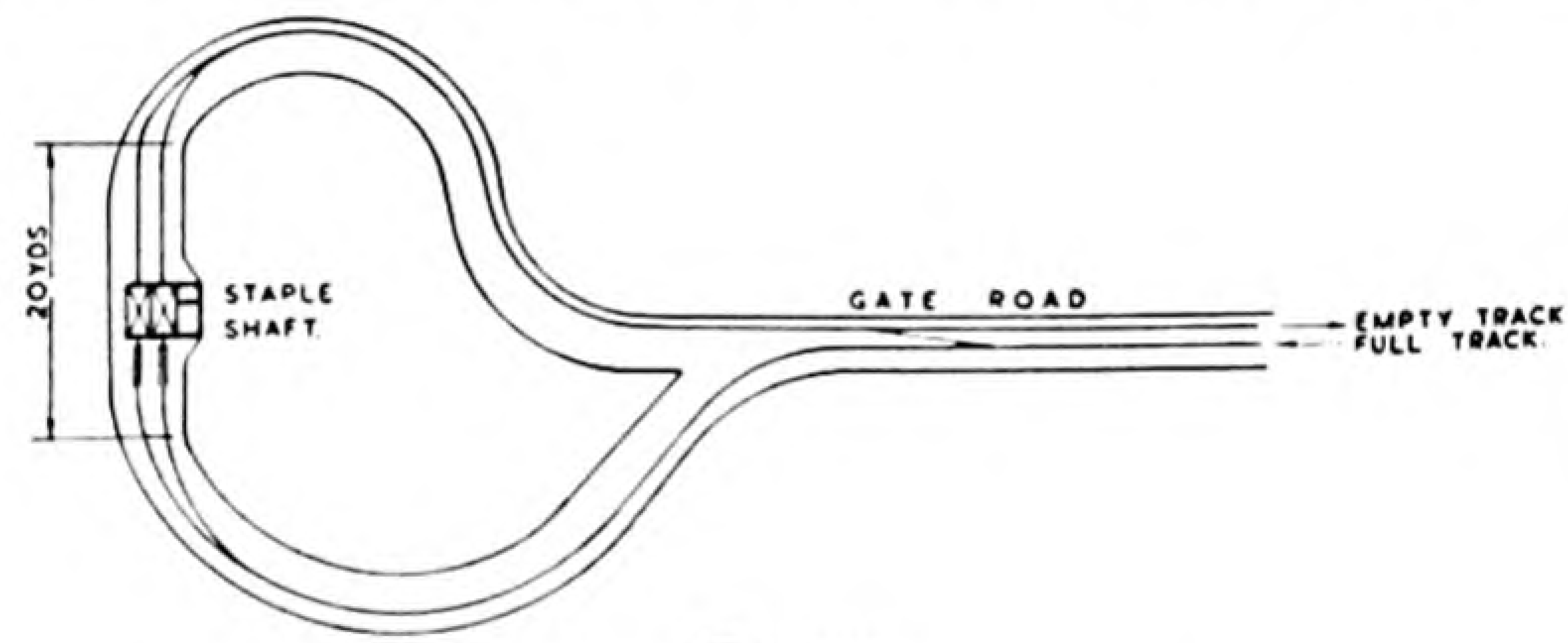
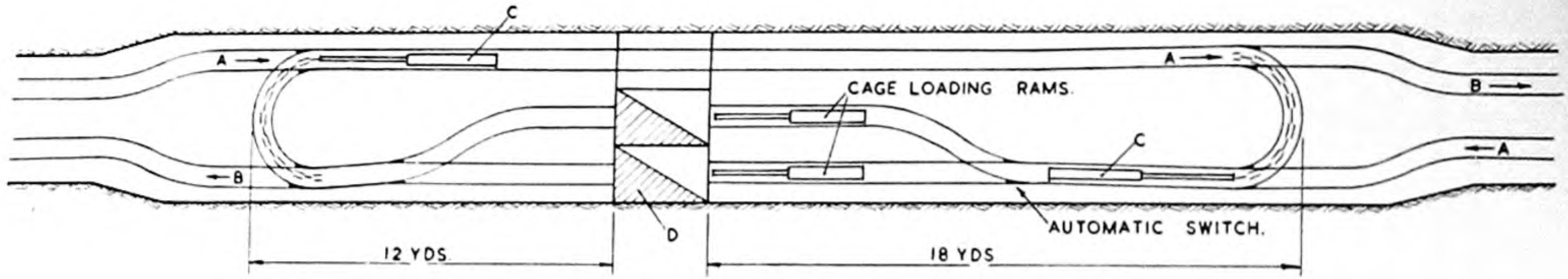


FIG. 290

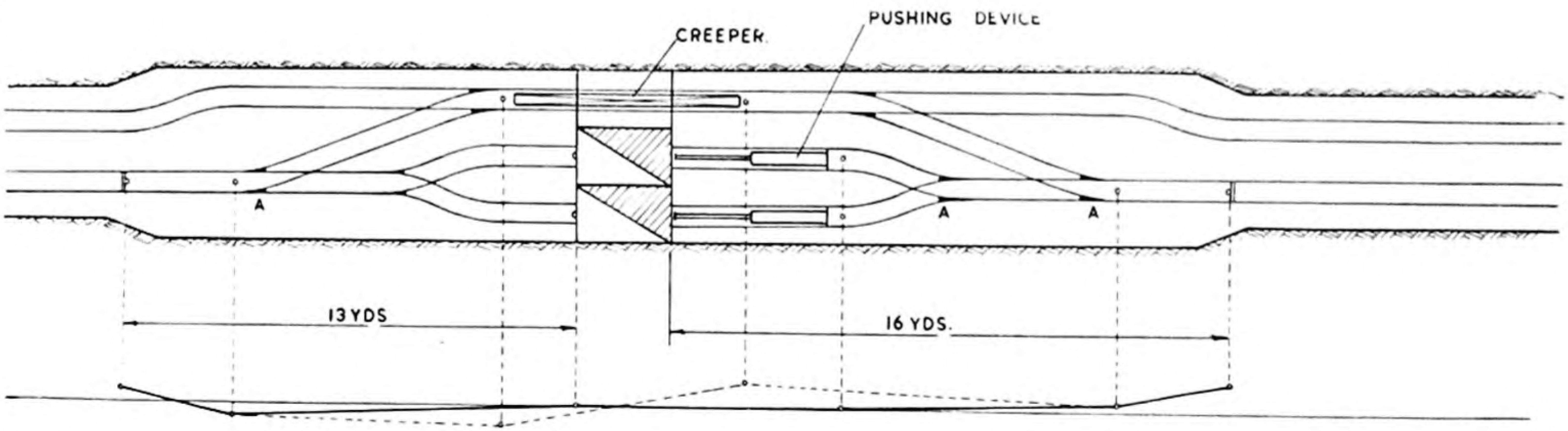
curve-pulling device operated by a ram. The full tubs from the right-side gate road run straight to the loading rams and the shaft. The empty tubs pushed out of the cage gravitate to the curve-pulling device and return via the small-radius curve and the track bypassing the staple shaft to the right-hand empty gate road. The full tubs arriving from the left side run past the shaft and are pulled round the small-radius curve to the decking rams. The empty tubs pushed out from the cages gravitate straight on to the left-hand empty gate-road track.

A similar layout is shown in Fig. 292, in which the short-radius curves and curve pullers have been replaced by automatic spring switches, and in which the track gradients are indicated. The maintenance of proper gradients is often difficult at staple-shaft stations because the level of the gate road is frequently altering due to coal working, and for this reason the layout shown in Fig. 291 is often preferred.



A. FULL TUBS.
B. EMPTY TUBS
C. CURVE PULLING DEVICE
D STAPLE SHAFT WITH 2 CAGE COMPTS. & ONE LADDER COMPT.

FIG. 291



A. AUTOMATIC SPRING SWITCH.

FIG. 292

If only one side of the shaft or one gate is being operated, only a single-wing layout along the same lines is necessary. The installation will be on the right side of the shaft if the left-side gate is operating, and vice versa. Since the roads behind the shaft are for tub circulation only, it may be necessary in some cases to have only single side operation and to deck out by hand.

Where a spiral chute is installed in the shaft and belt conveying is used for conveying the coal along the gate road, the direction of the shaft axis is of minor importance. The conveyor delivers the coal into the spiral-chute entry and, if the shaft axis is at right angles to the strike, i.e. to the direction of the gate road, a small bypass only is necessary for the tub circulation, as shown in Fig. 293.

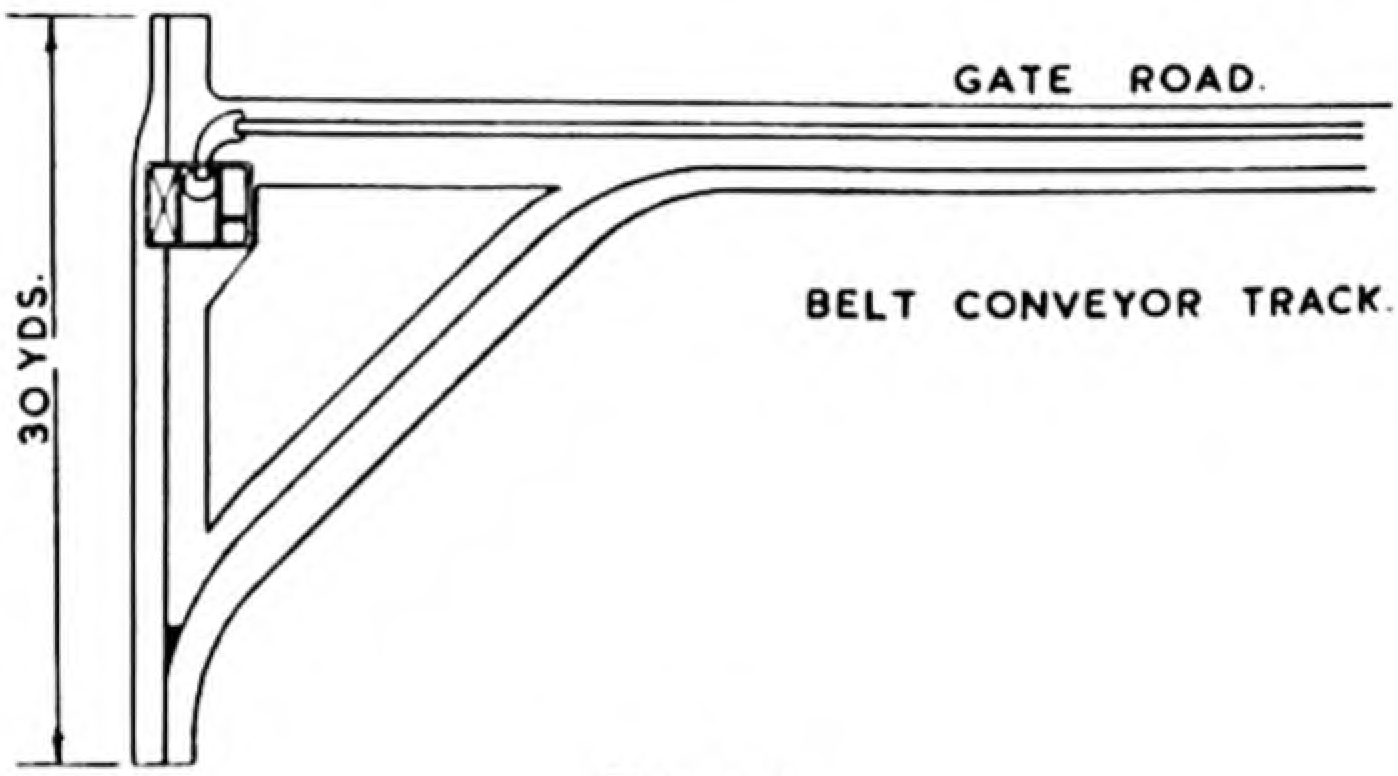


FIG. 293

(*b*) *Shaft-bottom layout with spiral chutes.* Where spiral chutes are used for coal transport in the staple shaft, the layout is considerably simplified, since the question of the location of the shaft axis with respect to gate road or main road does not arise. Apart from this advantage, the equipment described for cage hoisting is not required, since the coal is conveyed by belt conveyor along the gate road and single-cage winding with a balance weight is used only for materials transport. Tub loading takes place only at the shaft bottom. The layout in Fig. 294 shows a spiral-chute installation in which the spiral-chute cross-cut is at right angles to the main road. The distance from the spiral chute to the tub loading-point in the main road is about 8 yards, the coal being fed to the tubs by a short shaking conveyor.

(*c*) *Mechanical tub-handling equipment at the shaft bottom.* At staple shafts having only a small drawing capacity the circulation

and handling of the tubs can be effected by hand. The greater the capacity, the greater is the need to install mechanical tub-handling equipment, and at shafts and loading-points as much use as possible should be made of the equipment available. Rope haulage and decking rams may be used to bring the tubs to the loading-point and to load them into and out of the cages, but apart from this application in high-capacity shafts, mechanical decking is recom-

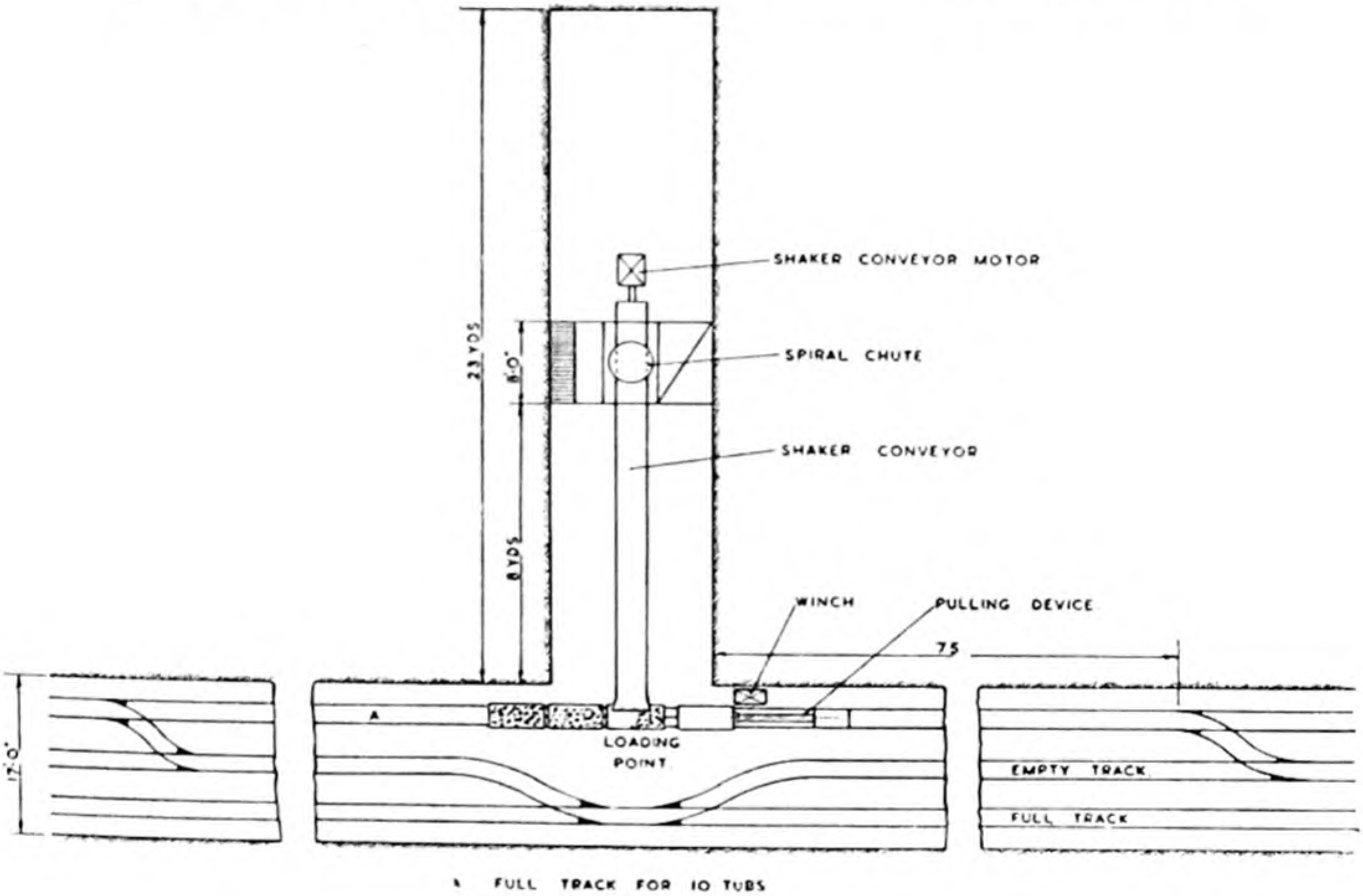


FIG. 294

mended. Under certain conditions, standard spring switches, acute-angle return-points and devices for pulling tubs round short-radius curves are necessary. Tilting platforms, always necessary in main-shaft Koepe winding, are only used very rarely in staple shafts; they may be necessary where a long rope is in use, as described in the case of a shaft-bottom winder installation in Part V, Sections 2 and 3.

Section 12. The Layout of Staple-shaft Bottoms and Sidings for Skip Winding

Several arrangements for the layout of the shaft bottom for

hoisting waste are illustrated in Fig. 295, (a) to (e). In most cases a single drum winder with a balance weight will be sufficient, the balance weight compensating the dead load and half the useful load. With this arrangement it is possible to have a shaft cross-section of about 65 square feet. A cage may be used in place of a balance weight to serve for man-riding and materials transport. In both cases the

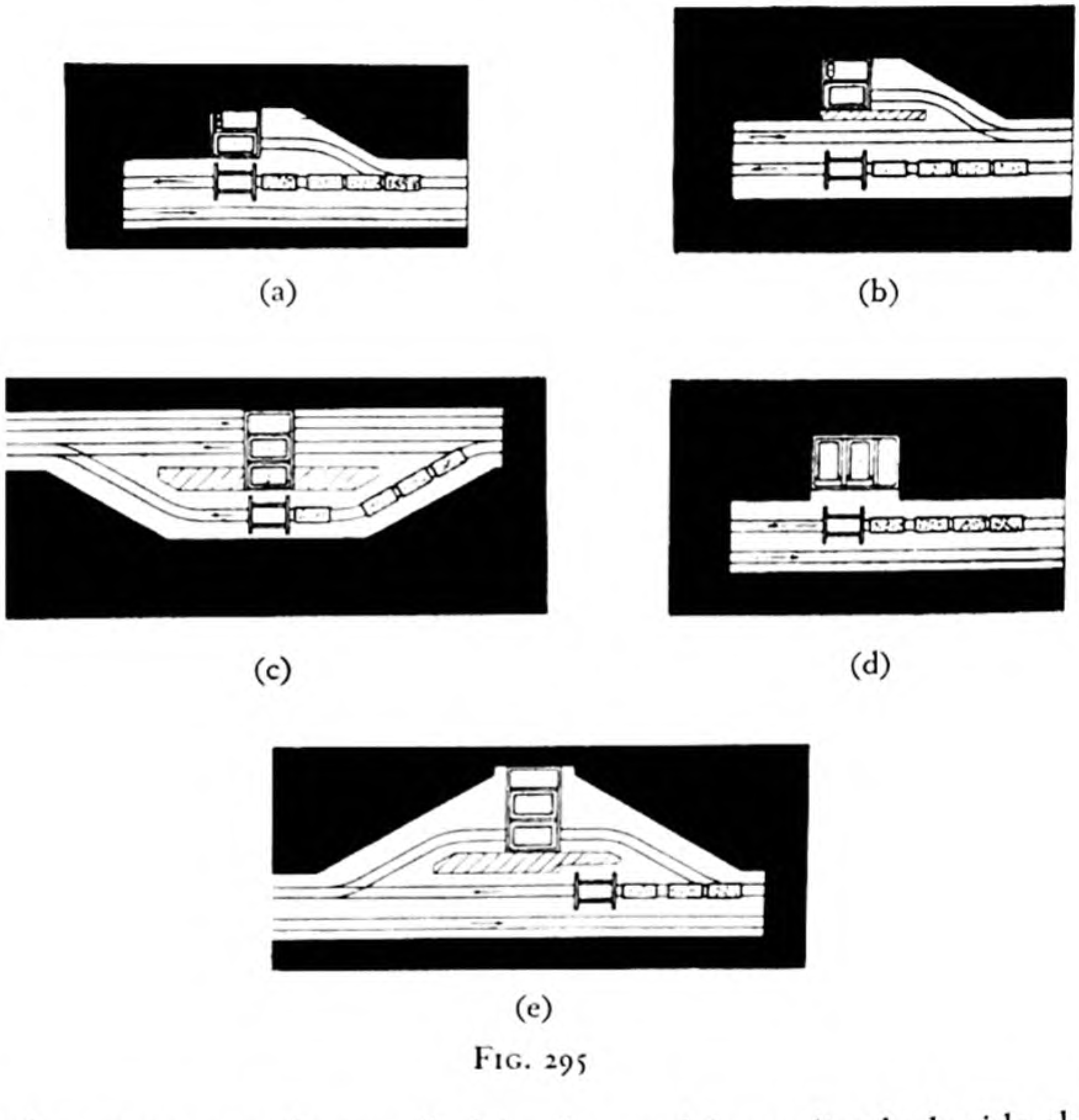
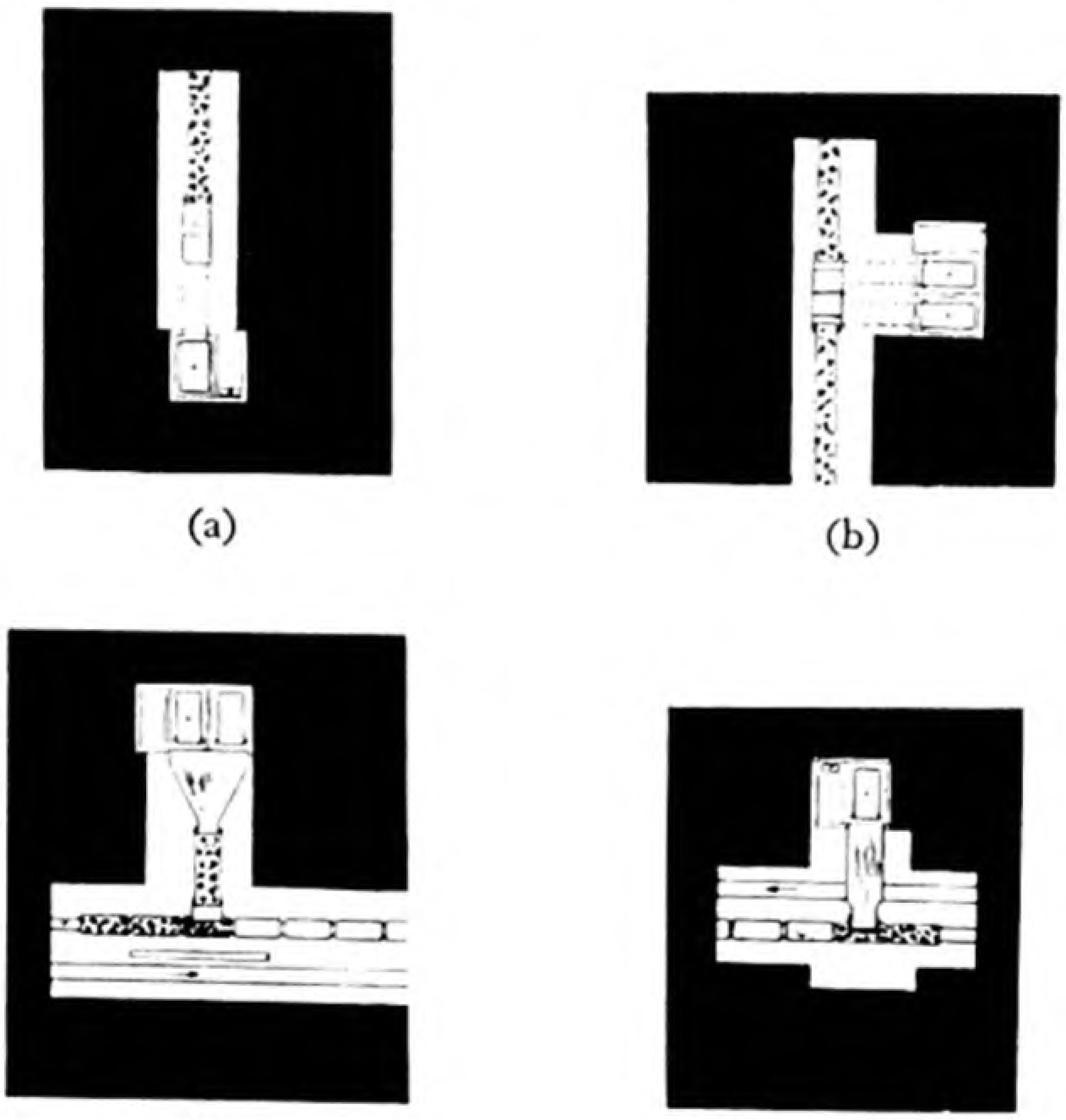


Fig. 295

tipplers are generally installed in the road immediately beside the skip compartment. Side filling is usually adopted, and in the case shown in Fig. 295 (a) the shaft is on the full track side of the road. If the staple shaft is located on the empty side of the road, the arrangement will be shown in Fig. 295 (b). Another possible variation occurs when the full track is in line with the skip compartment in the shaft, in which case the tippler track is run into a bypass and the empty track runs over the ladder or man-way compartment shown in Fig. 295 (c).

Where a double-drum winder is being used with two skips, the layout depends on the alternative means, if any, for materials transport. If this is not required, then the layout may be as shown in Fig. 295 (d), in which end-on loading is used. Where materials transport must also be carried out in the same shaft, the tippler is set at the side of the staple shaft as in Fig. 295 (e), so that both skips may be loaded on the narrow side.

Examples illustrating the layout for loading and unloading sta-



(a) (b) (c) (d)

FIG. 296

tions in the case of coal hoisting are shown in Fig. 296(a) to (d). The loading-station for a single-end installation is illustrated in Fig. 296 (a), in which belt-conveyor transport is used, while Fig. 296 (b) shows the alteration required for double-sided delivery by belt conveyor to the skip-filling pocket. Where the full track is on the shaft side of the roadway, the layout illustrated in Fig. 296 (c) must be adopted, while Fig. 296 (d) shows the arrangement where the full track is on the side of the road remote from the shaft. In the latter case, which is preferable, the operator controlling the loading is placed in a niche on the far side of the road.

PART III

TRANSPORT IN LEVEL MAIN ROADS

Section 1. *Diesel Locomotives*

In British coal-mining, the diesel* locomotive takes prior place at the present time for level main-road haulage, while in the Ruhr 40 per cent. of approximately 2,500 locomotives in use are of the trolley-wire type. In the Ruhr, the diesel locomotive and battery and compressed-air-operated locomotives are used for gathering tubs, gate-road transport and main-road haulage. For main-road haulage, the German diesel locomotives are constructed in a range of from 30 to 90 horse-power or more and weigh from 7 to 11 tons. For gate-road haulage, the engines are either of the horizontal single-cylinder type or the six-cylinder block type, working on the four-stroke cycle with solid injection. The engines run at between 800 and 1,200 r.p.m. and are water-cooled by natural circulation in the single-cylinder type, but by forced circulation in the multi-cylinder engine.

All moving parts of the locomotive must be protected against dust and water; the engine and transmission gear are totally enclosed, and these, together with the accessories, are further covered with steel plates provided with cooling vents.

The engine may be started by compressed air transported in cylinders on the locomotive. The cylinders can be filled either by the engine itself or by a separate compressor.

The diesel locomotive has certain hazards which have to be guarded against. These are the possible ignition of a methane-air mixture (fire-damp) and the production of carbon monoxide in the exhaust fumes. It is, however, possible to remove these hazards by incorporating devices into the engine design and by maintaining a high standard of ventilation in the main roads where the locomotives are in use. The fuel-oil tank must be fixed rigidly on the locomotive frame and protected against damage. The use of taps or valves for controlling and draining fuel oil should be avoided, and the feed pipe must be leak-proof and arranged so that oil cannot drip on to

* These engines are solid-injection oil engines and not air-injection engines as in the true diesel. Usually they are referred to as compression-ignition (C.I.) engines, but the description 'diesel' is used throughout the book according to common usage in underground locomotive practice.

hot parts of the exhaust system. The exhaust cooling system is especially important and at no point should the surface temperature exceed 200° C., at which temperature it is possible to ignite a coal dust-air mixture. For this reason, cooling water is injected into the exhaust pipe, and this system also effectively assists in washing the engine exhaust gas. A water-tank behind the exhaust pipe serves the same purpose. The volume and temperature of the water in the tank is kept constant by a water-injector and the exhaust gas is cooled so that the temperature at which it is exhausted into the mine roadway does not exceed 70° C. Should the water-supply to the injector cease due to lack of water or blocking in the water-filter, the engine is stopped automatically by a control device operated by the water-injector. Fire hazards require that equipment be included for quenching any fire in the interior of the locomotive engine cover by filling it with carbon dioxide by a control in the locomotive cab. A portable fire-extinguisher also should be carried.

The danger of the ignition of fire-damp is prevented by suppressing any sparks or flames from the exhaust with a flame-trap and a similar protection on the air-suction intake.

The supervision of the locomotives in operation requires special care to keep the CO-content of the exhaust gas below the prescribed maximum. The regulations governing the use of diesel locomotives in German mines prescribe a minimum air quantity of 160 cubic feet per minute per locomotive horse-power in all roads travelled by diesel locomotives.

Diesel locomotives of flame-proof design and of British manufacture have been in use in British coal mines since 1939 when a 25-h.p., $4\frac{1}{2}$-ton, four-wheeler locomotive, running on a 2-feet-gauge track was installed at Rossington Colliery for a man-riding haulage. The important manufacturers of locomotives such as Hunslet, Ruston, Huwood-Hudswell and North British have since largely contributed to the growing numbers of diesel locomotives in use for both coal and man haulage.

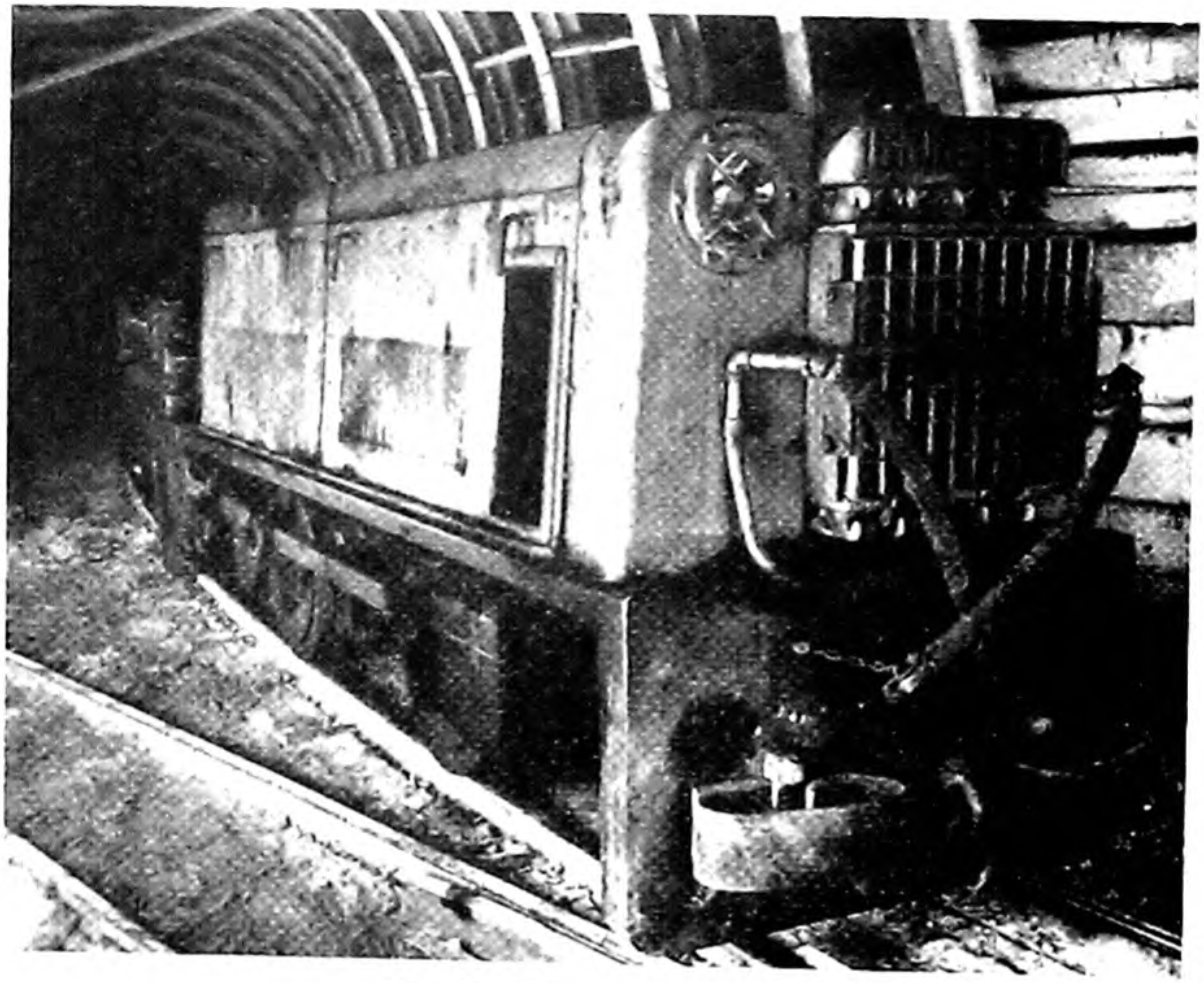
The Hunslet Engine Co. Ltd. have now a range of five flame-proof diesel locomotives, of which the 16/24-b.h.p. 'pit pony' has already been briefly described. The 45-, 65- and 100-b.h.p. locomotives are powered by Gardner engines, while the 70-b.h.p. locomotive has a Meadons engine. The whole range of locomotives are built to standards fully equal to those adopted for main-line

locomotives in order to overcome the arduous conditions of pit work. The 70- and 100-b.h.p. locomotives are shown in Fig. 297. The 70-b.h.p. model is a 10-ton, 15-m.p.h. locomotive for general-purpose duties and has a four-speed gear box. The 24-mesh wheels are spread over a base of 4 feet 3 inches. The length over the buffer beams is 12 feet 9 inches, maximum height 5 feet 3 inches and overall width 3 feet 10 inches. The locomotive is supplied for all gauges from 1 foot $11\frac{1}{2}$ inches to 3 feet 6 inches, and can traverse curves of 50-foot radius. Track speeds are $4\frac{1}{4}$, $6\frac{1}{4}$, $9\frac{1}{2}$ and 15 m.p.h., and tractive effort in bottom gear is 5,250 lb. The minimum rail weight is 28/30 lb. per yard. The larger 100-b.h.p. locomotive is a 15-ton, 15-m.p.h. six-wheeler for heavy coal haulage or man-riding, particularly over severe gradients. The 24-inch wheels are spread over a base of 5 feet 3 inches. The length over the buffer beams is 14 feet 3 inches, maximum height 5 feet 6 inches and overall width 4 feet 6 inches. The locomotive is built for all gauges from 2 feet 6 inches to 3 feet 6 inches and can negotiate curves of 60-foot radius. The track speeds and minimum rail weight are the same as for the 70-b.h.p. locomotive, while the tractive effort in bottom gear is 8,000 lb.

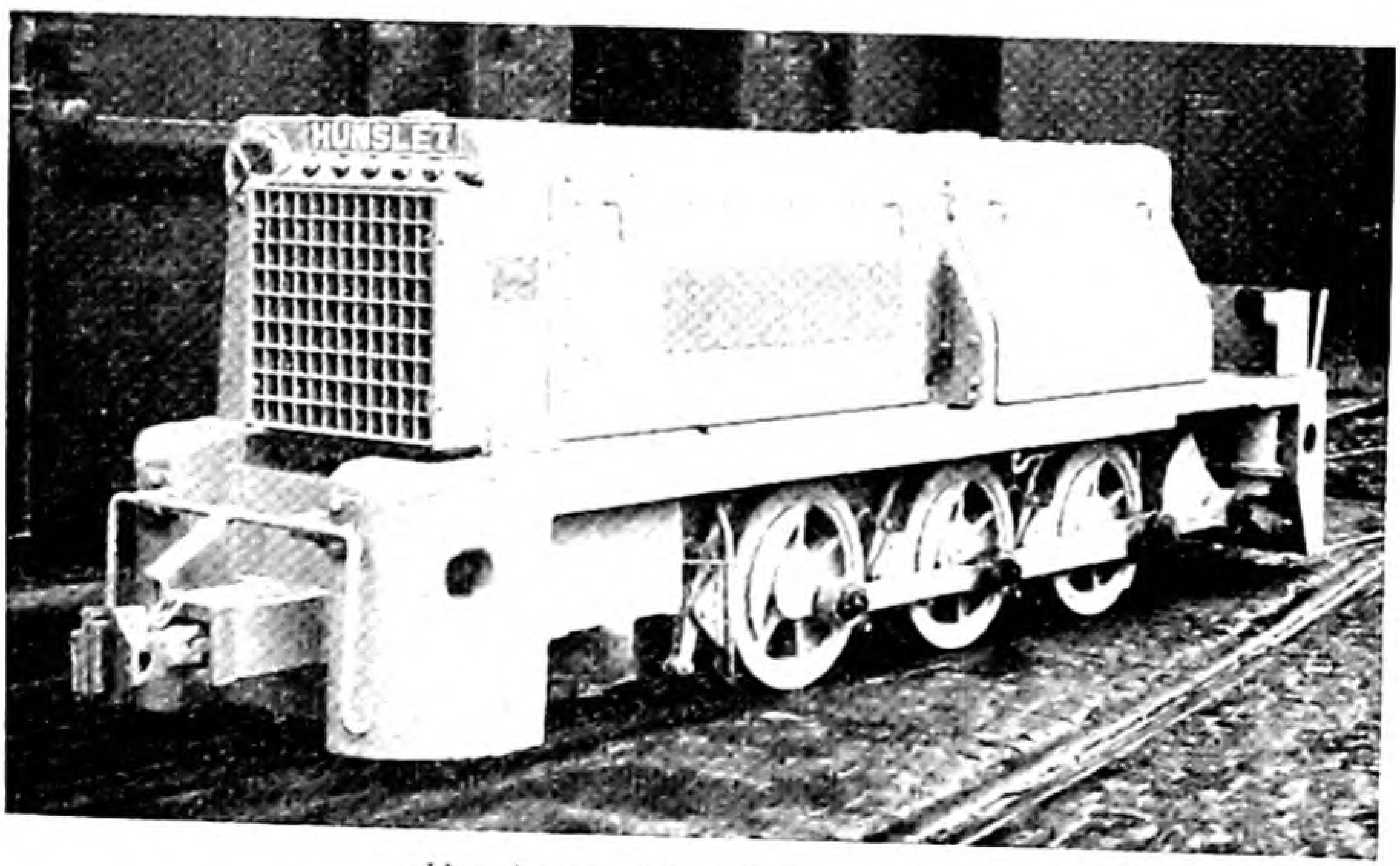
The Hunslet patent exhaust-gas conditioner is combined with the flame and spark arrester which is shown individually in Fig. 298. The whole of the conditioner is made in stainless steel and it is provided with automatic equipment so that in the event of the water level falling below a certain point the fuel is automatically cut off from the engine and the brakes applied. The locomotives are supplied with spare arresters which are changed on the exhaust side every shift. The main clutch is a patent, air-operated single-plate type, while the speed changes are operated by a pre-selective automatic gear-change mechanism coupled with the main clutch and throttle.

A powerful hand-brake is fitted, but in service, braking control is by compressed air coupled up to the whole train set in man-riding, so that every carriage is fitted with power-brake equipment operated by the locomotive driver. An emergency feature is also incorporated which immediately applies all brakes in both sections of the train in the event of a breakaway.

Ruston and Hornsby Ltd., who installed a diesel locomotive underground in a gypsum mine in 1932 and were granted the first



Hunslet 70-b.h.p. Locomotive



Hunslet 100-b.h.p. Locomotive

Fig. 297

Buxton certificate for locomotives for use in gassy mines, have a range of six locomotives for underground work. This range covers 20-, 30-, 40-, 48-, 75- and 100-b.h.p. locomotives. The latest

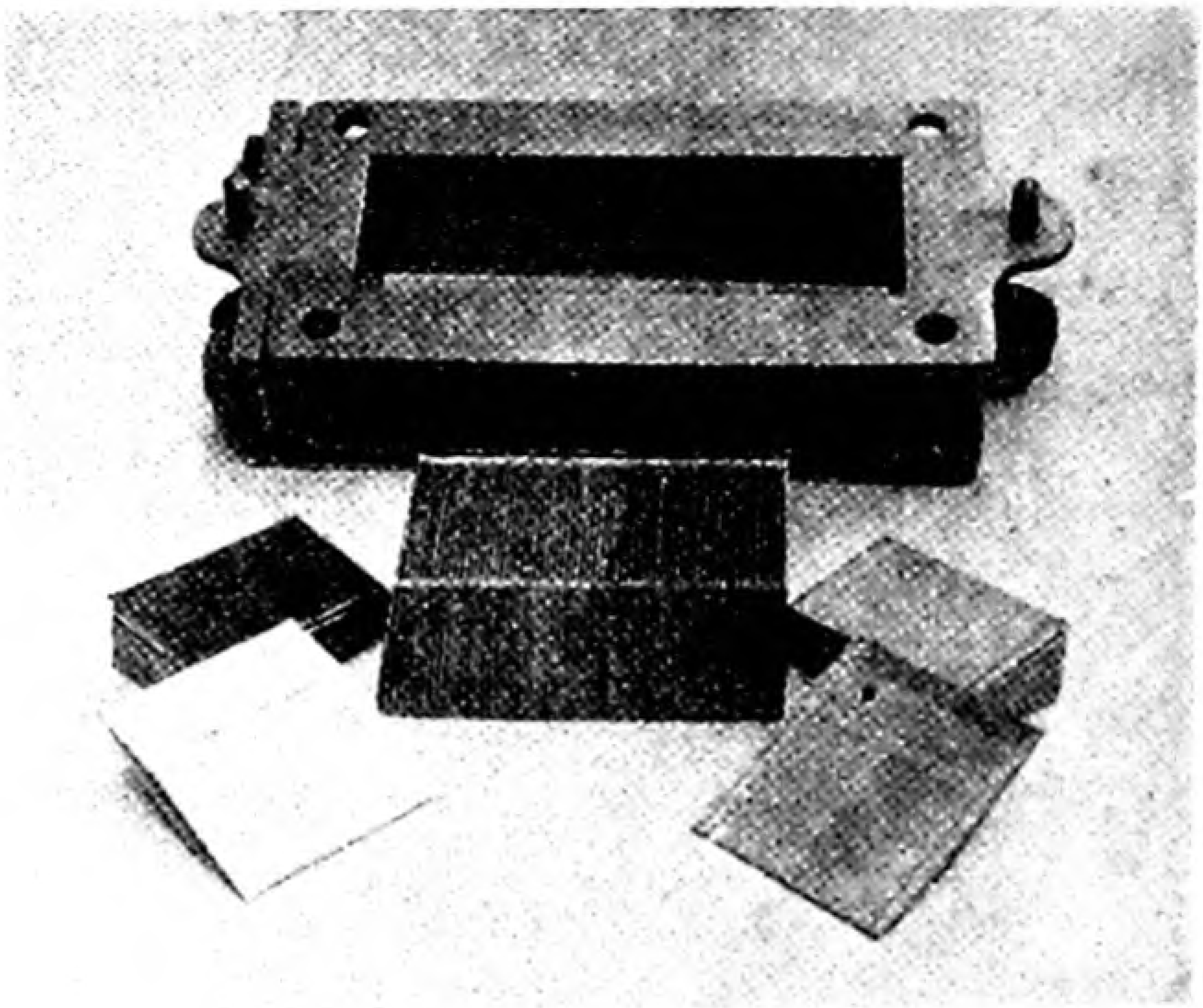


FIG. 298

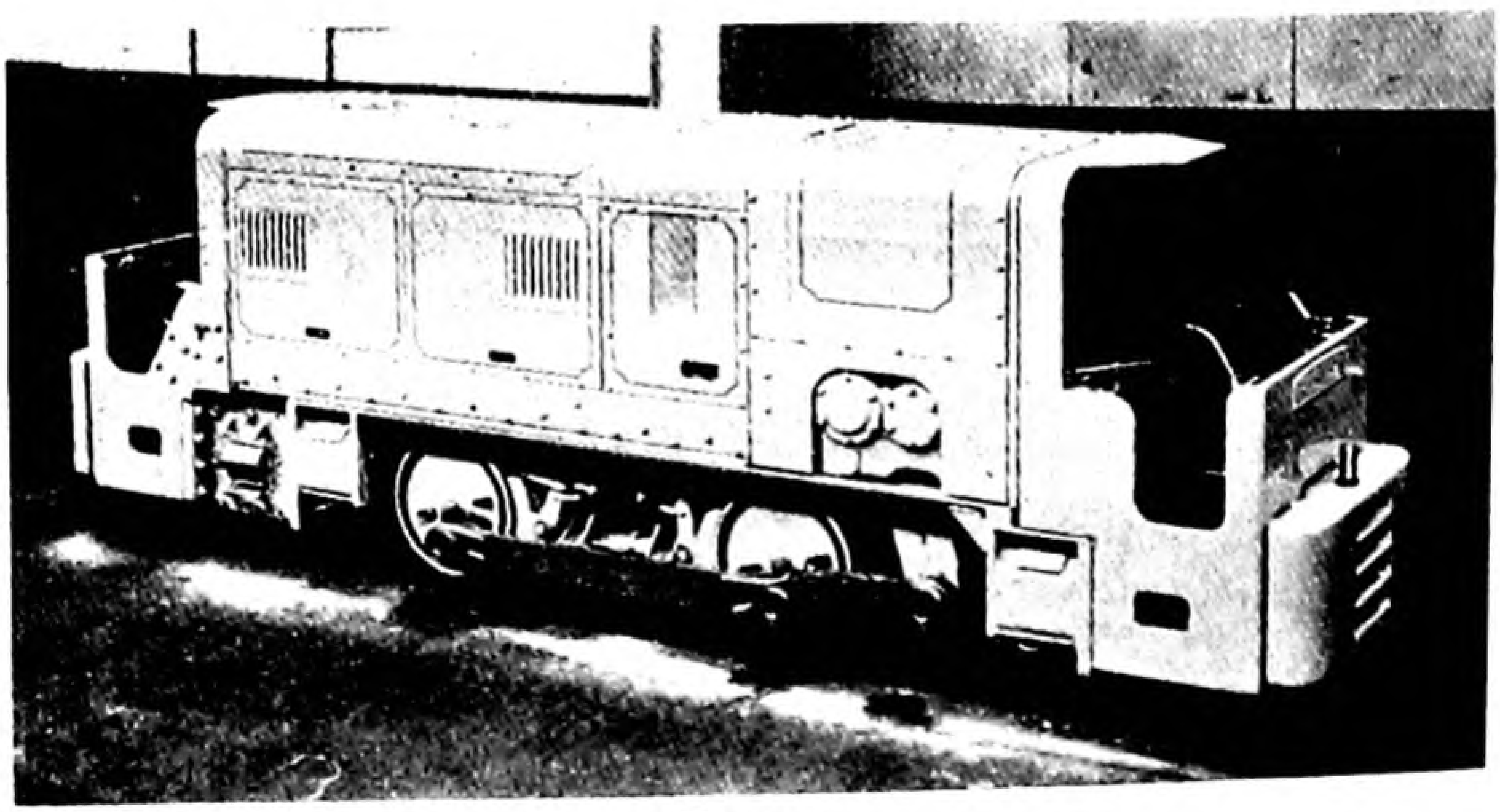


FIG. 299

addition to this range, the 75-b.h.p. locomotive, has several unique features, in particular the provision of a driver's cab at each end, thus giving complete visibility. This locomotive is shown in Fig. 299, while the 7-ton, 48-b.h.p. locomotive, fitted with Westinghouse straight air-brake equipment, is shown in Fig. 300. The 75-b.h.p.

locomotive is built on the unit-construction principle, each unit, i.e. engine radiator, gearbox, etc., being detachable without affecting adjacent units. The locomotive can be supplied for any gauge from 1 foot $7\frac{5}{8}$ inches to 3 feet 6 inches and will negotiate a 30-foot curve. The normal method of starting is by low-pressure compressed air through an air starter motor, the compressed air being supplied to a receiver from a compressor driven from the main engine. The exhaust conditioner is readily accessible for cleaning and inspection

FIG. 300

while flame-traps are fitted on both the exhaust and outlet and air inlet to the engine. All Ruston locomotives for coal mines are fitted with electric lighting on a direct system from an enclosed dynamo, all this equipment being flame-proof. The locomotives are powered by Ruston compression-ignition engines with flexible couplings between the engine and the gearbox, which is of the constant-mesh type. The 100-b.h.p. locomotive is provided with a scoop-controlled fluid coupling, while the gearbox is an S.S.S. (Synchro Self Shifting) Powerflow unit giving three speeds in both forward and reverse directions. There are no foot controls on the locomotives, a single lever only being provided. Each gear has its own separate clutch and when the lever is moved, one clutch is disengaged and another engaged.

The Huwood-Hudswell 100-b.h.p. diesel locomotive was the first of this horse-power to be built. The Huwood-Hudswell combination also produces a 68-b.h.p. locomotive which is similar in general design and in many of the details, such as transmission, brake components and radiator elements, to the 100-b.h.p. locomotive. The engines in both types are Gardner oil engines running at 1,700 r.p.m., the 68-b.h.p. and 100-b.h.p. being four-cylinder and six-cylinder versions. The makers guarantee an extremely low fuel consumption of 0·37 lb. per brake horse-power per hour under full load. The locomotives are fitted with a conditioner box on the exhaust, in which the gases leaving the cylinders are cooled by a spray which serves both to cool the gases and replenish the water in the conditioner box. This box is fitted with internal baffles and half-filled with water containing soda-ash to neutralise the acid effects of the gases. Details of the spray and the conditioner box are shown in Fig. 301. The cooling and replenishing water is gravity fed from a tank forming a section of the bonnet top to a small rotary pump which is belt-driven from the fan shaft. The pump delivers water at a pressure of 60 lb. per square inch to the nozzle in the exhaust pipe in the form of a very fine spray. The tank supplying the spray water is fitted with a float which operates a cock placed in the fuel-supply pipe. Should the water fall below a predetermined level, the fuel supply is shut off and the engine stops. The general arrangement of the 100-b.h.p. locomotive is shown in Fig. 302, which indicates the location of the exhaust sprayer, conditioner box and exhaust and inlet flame-traps. The 100-b.h.p. locomotive adopts the Vulcan-Sinclair fluid coupling manufactured by the Hydraulic Coupling & Engineering Co. Ltd., together with the Sinclair S.S.S. Powerflow gearbox. The fluid coupling may be rapidly disengaged when the engine is brought to idling speed. The 100-b.h.p. 15-ton locomotive is of the six-wheel type and can be supplied for gauges from 1 foot 11 inches to 3 feet 6 inches and can negotiate curves of 55-foot radius. It has a wheel base of 5 feet 6 inches and a length over buffer beams of 15 feet 11 inches, a height of 5 feet 6 inches and a normal width of 5 feet. It has three speeds in each direction of 3·8, 7·3 and 14 m.p.h., at which tractive efforts of 8,300, 4,360 and 2,280 lb. are developed. The 68-b.h.p. locomotive, which is 10 tons weight, has a length of 14 feet 5 inches over buffer beams, 5 feet 3 inches high and from 4 feet 10 inches to 3 feet 4

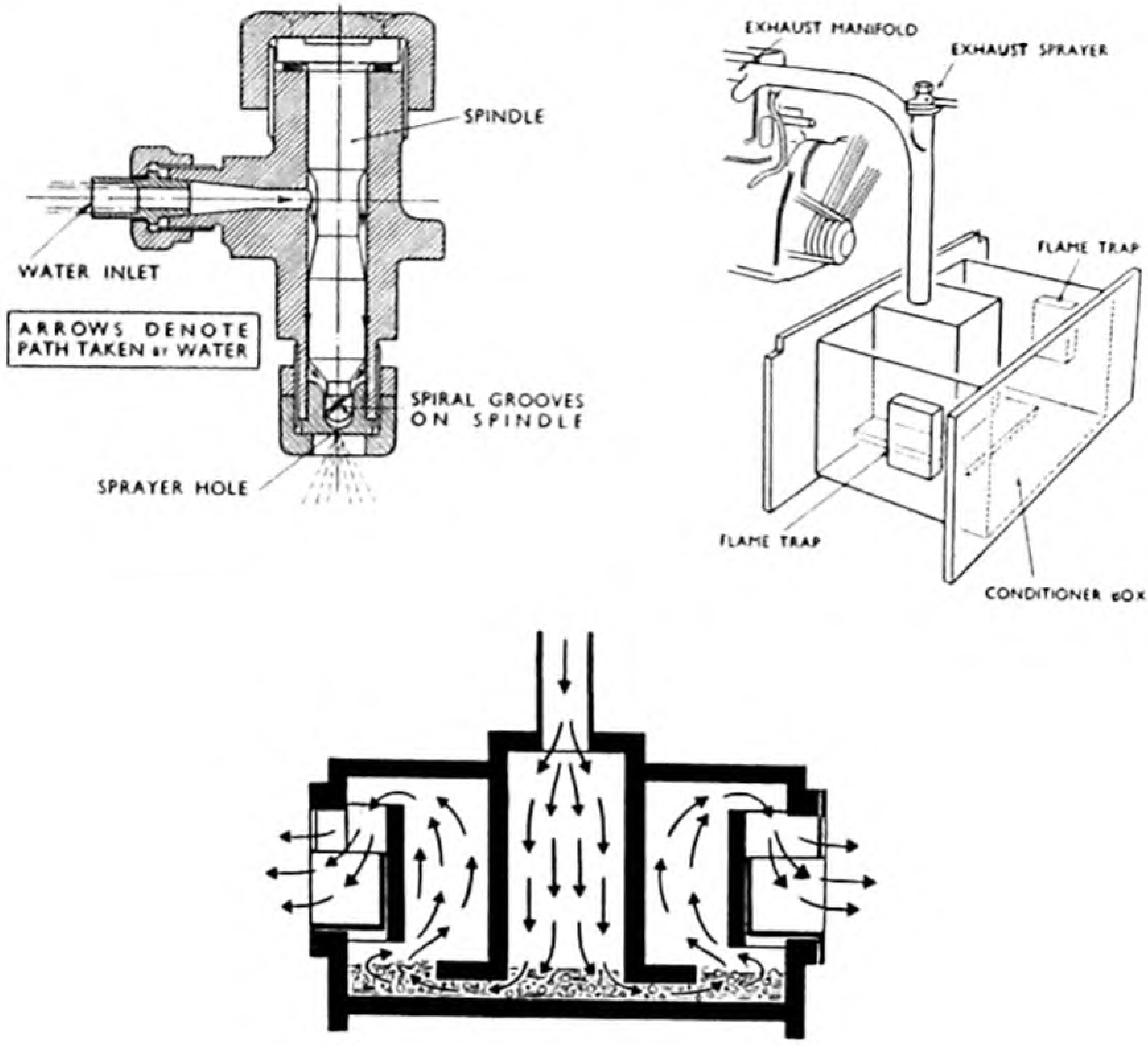


FIG. 301

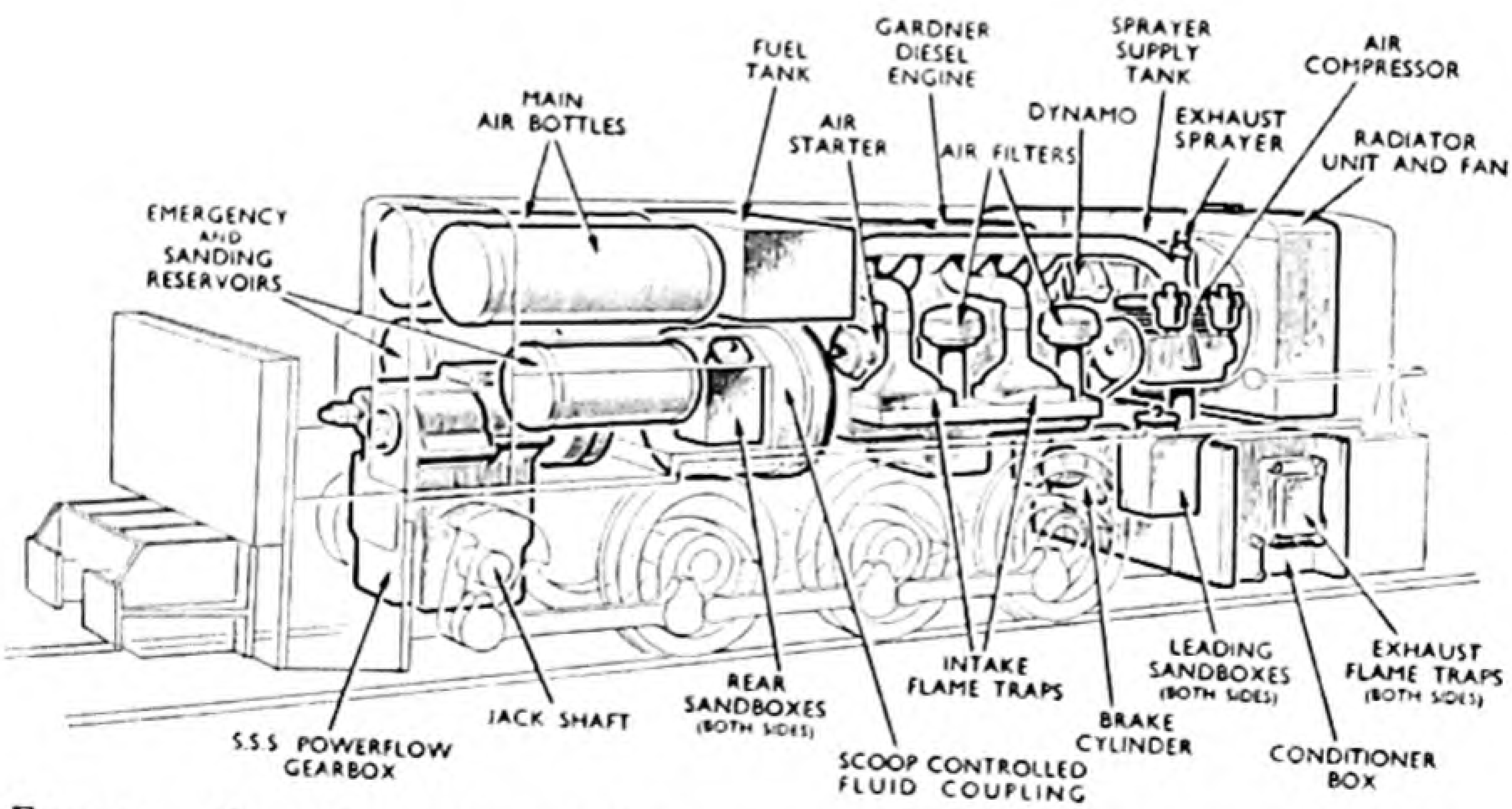


FIG. 302.—General arrangement of main components Huwood-Hudswell 100-b.h.p. mines type diesel locomotive.

inches wide, depending upon the gauge. The speeds on the three gears are the same as for the 100-b.h.p. locomotive, and the maximum tractive effort developed is 5,260 lb. at 1,700 r.p.m. The 100-b.h.p. locomotive is shown in Fig. 303 and the 68-b.h.p. in Fig. 304.

The North British Locomotive Company introduced a 100-b.h.p. diesel locomotive in which the Voith-North British turbo transmission has been adopted. This transmission consists of two hydraulic circuits comprising a torque converter for starting and

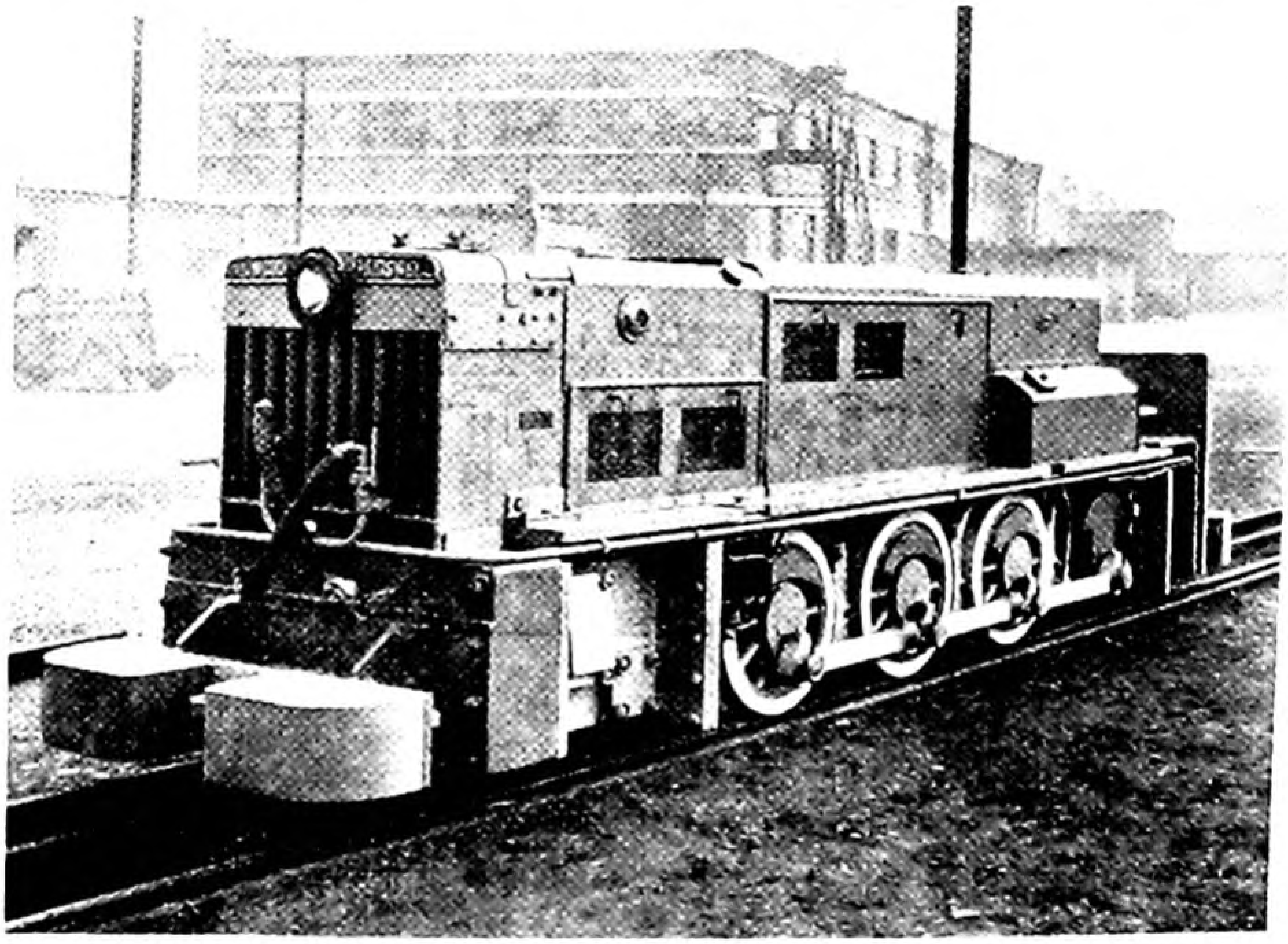


FIG. 303.—Huwood-Hudswell 100-b.h.p. locomotive.

low speeds and a fluid coupling for higher speed. There is no gear changing or clutch operation. The torque converter and fluid coupling are self-contained and are filled or emptied automatically by a governor actuated by the speed at the rail and the resistance encountered. Control by the driver is confined to the throttle lever only. The locomotive is shown in Fig. 305 in which the driver's cab is clearly seen. There is no mechanical connection between the engine and road wheels, thus shock loads from the engine or track are eliminated. The engine is usually a Paxman 6 RQE, six-cylinder, four-stroke cycle compression-ignition oil engine, developing 100 b.h.p. at 1,250 r.p.m., but a Crossley BWL 5,

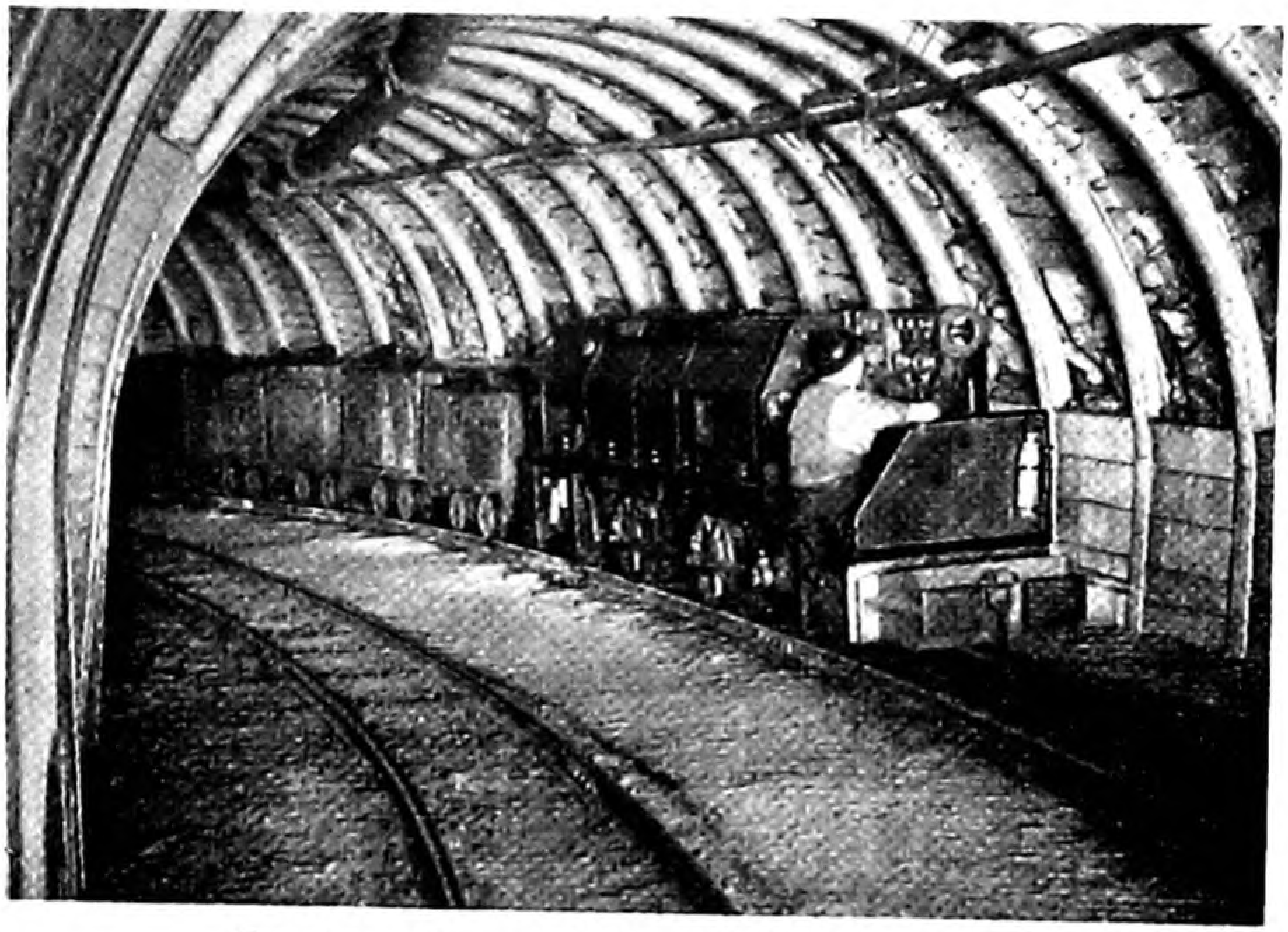


FIG. 304.—Huwood-Hudswell 68-b.h.p. locomotive.

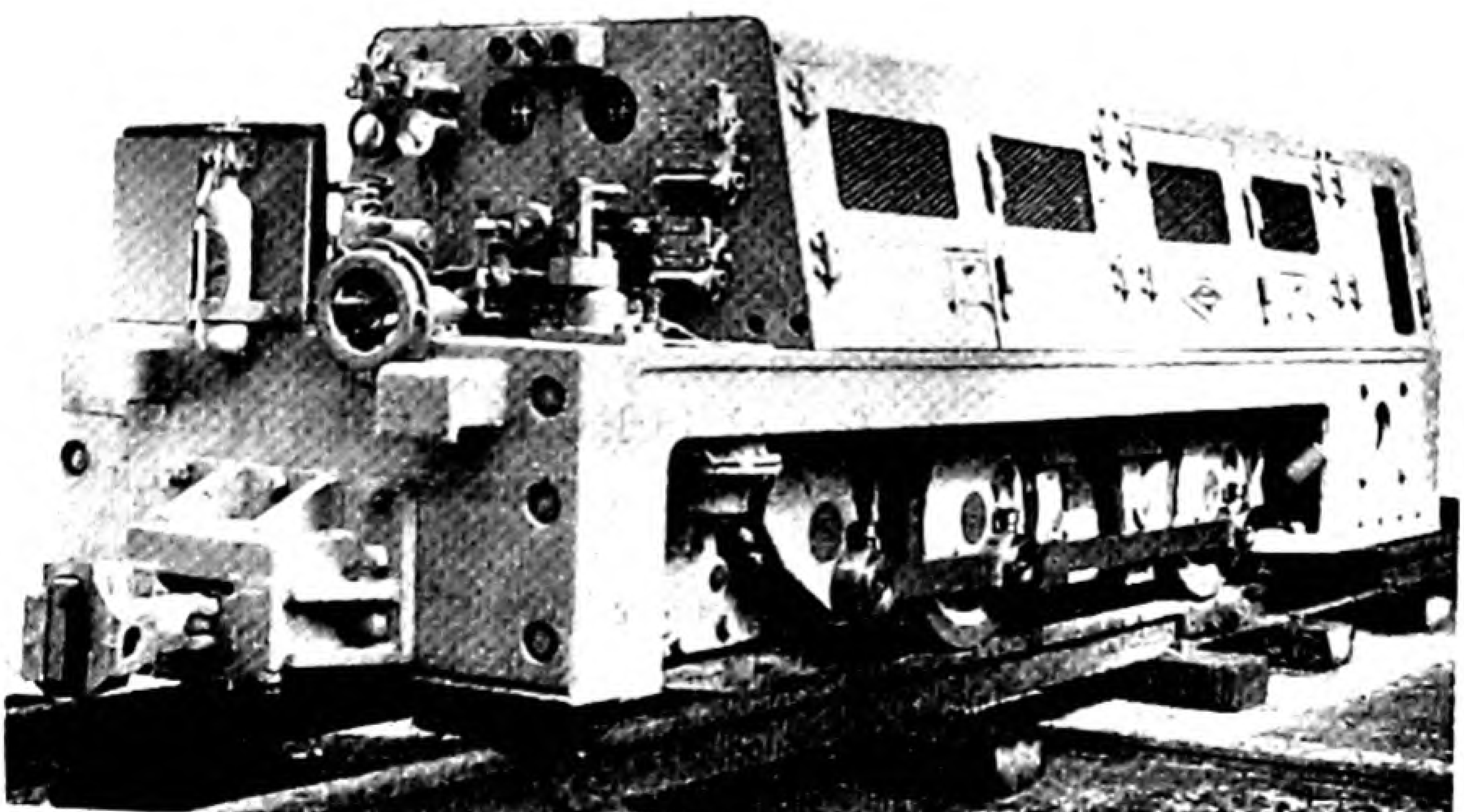


FIG. 305

five-cylinder, four-stroke engine giving the same output at the same speed, can be fitted as an alternative. The tractive effort at 25 per cent. adhesion is 8,400 lb. at sea-level. The locomotive which weighs $14\frac{1}{2}$ tons and can run at 15 m.p.h., can be supplied

IO TON DIESEL LOCOMOTIVES

TRACTIVE EFFORT-SPEED CURVES

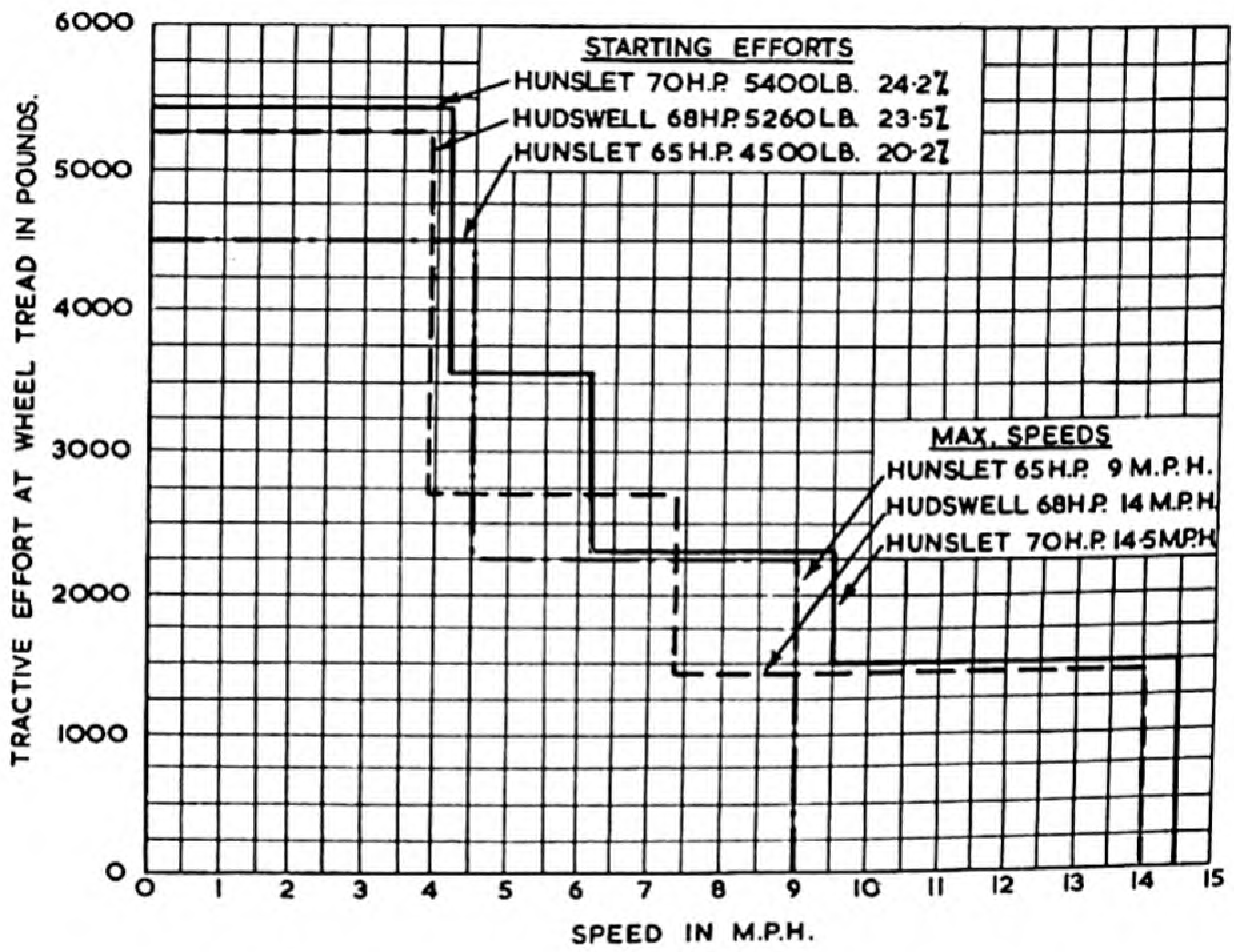


FIG. 306

for track gauges from 2 feet $5\frac{1}{2}$ inches to 3 feet 6 inches. The minimum radius of curve which the locomotive will negotiate is 45 feet.

Comparative tractive effort-speed curves given by Green* for the 10-ton locomotives and the 15-ton locomotives are shown in Figs. 306 and 307.

Haulage costs with diesel locomotives. The initial capital cost for diesel-locomotive installations is less than for any other type of

* 'Underground Locomotives', T. E. Green, N.C.B. Summer School, Cambridge, 1949.

equal capacity. In addition to the locomotives, only mobile fuel-oil tanks are required as main items of additional equipment, the cost of which is low, although special accommodation is required when refuelling. The cost of the locomotives per horse-power depends on

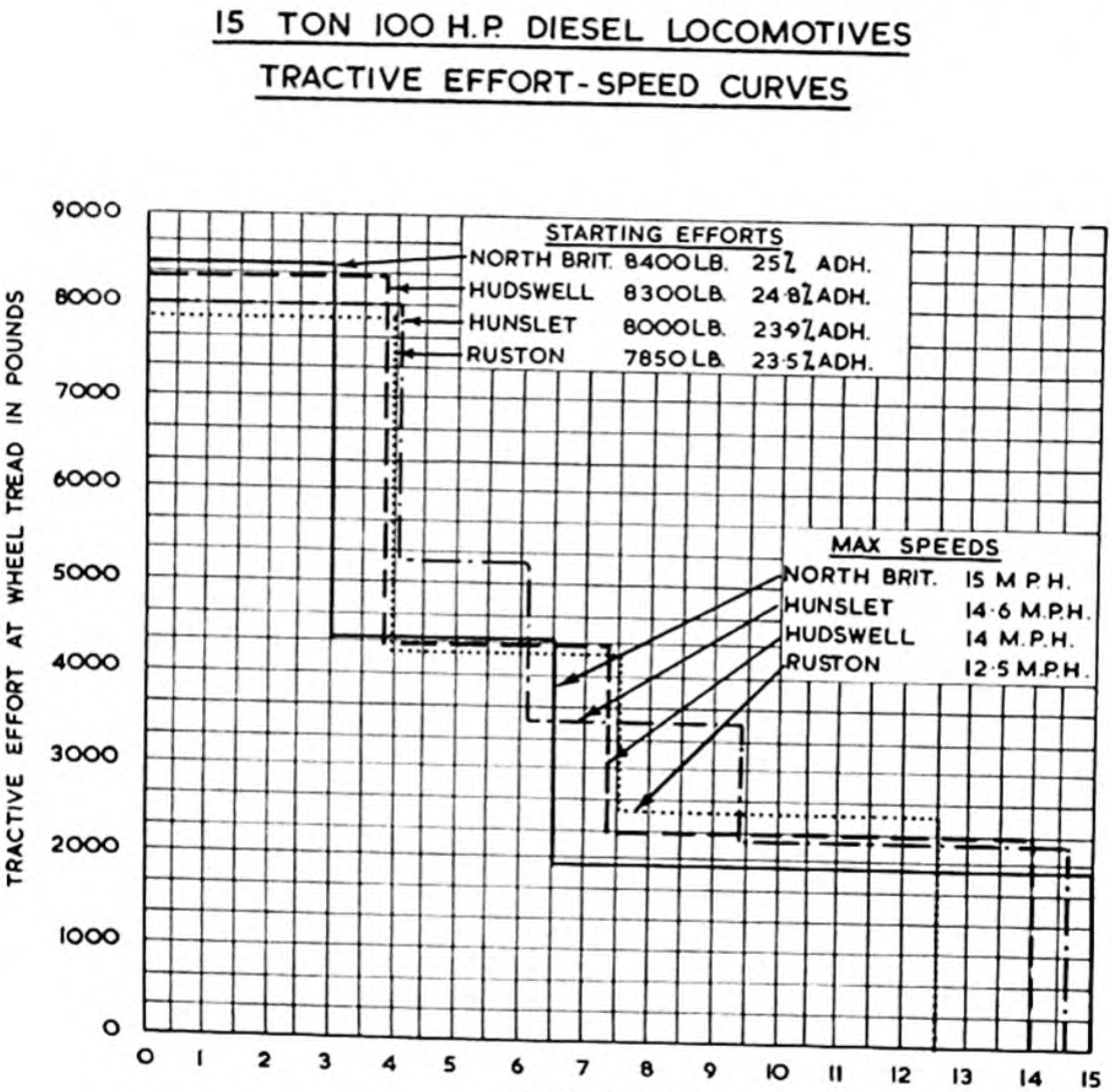


FIG. 307

the design, construction and power. As with other types of locomotive, the cost per horse-power decreases above a certain power, e.g. for a 50-h.p. locomotive the cost is about £75 per horse-power, for a 70-h.p. locomotive about £70 per horse-power and for a 100-h.p. locomotive about £65 per horse-power. The number of reserve units required will depend upon the total number of locomotives in operation. Where 10 locomotives are in use, 2 or 3 or from 20 per cent. to 30 per cent. in reserve are sufficient, while the additional reserve required for a larger number of locomotives in

operation may be from 15 per cent. to 20 per cent. For ease and speed of maintenance with, say, 5 locomotives in operation, the stores should include the following: 1 engine, atomisers, 1 pair of wheels and axle, 1 fuel-oil pump, 1 injector water-pump, 1 radiator, 1 exhaust baffle and several sets of baffle plates, etc.

If the reserve factor for 10 locomotives is considered to be 1·3, in the case of a larger number of locomotives in operation the reserve factor can be reduced to from 1·15 to 1·2. The initial capital cost for diesel locomotives ready for operation can be expressed as:

$$C = (1{\cdot}3 \text{ to } 1{\cdot}15)\, a \times n \times \text{h.p.}_{\text{Loc}}\ \pounds,$$

where a is the cost per horse-power, n is the number of locomotives in operation, and $\text{h.p.}_{\text{Loc.}}$ is the horse-power per locomotive. Thus, for 10 locomotives in operation the initial capital cost would be $1{\cdot}3 \times \pounds 50 \times 10 \times 50$, or about £35,000.

The operational cost should be based upon a locomotive life of 10 years, which is about half the life usually taken for other types. The wages cost per locomotive per day can be based upon 2 drivers and from 1 to 1·5 shifts for maintenance. The consumption of spares is usually greatest for the frame, driving gear, brake blocks, tyres and safety appliances. Taking a list of 10 years, the average total maintenance cost per locomotive per annum can be assumed to be £350. The average consumption of fuel oil is about 0·05 gallon per locomotive horse-power per hour, which will vary according to the condition of the mine cars and track and the locomotive horse-power, but can be assumed to be about 0·006 to 0·009 gallon per gross ton-mile. Taking all factors into account, the depreciation cost will be less than for trolley-wire or battery locomotives, but the maintenance and power costs are higher. The total operational cost is approximately the same as for battery locomotives, but higher than that for trolley-wire locomotives.

Section 2. Battery Locomotives

This type of locomotive has the battery as the source of power and has a similar track mobility to the diesel locomotive. The battery, which must have an adequate capacity to cover a reasonable working time before changing, has a comparatively short life. The motors and control equipment are the normal flameproof traction type. Locomotives suitable for main-line haulage range from 7 to 12 tons

in weight, have a nominal horse-power of from 35 to 90 and operate on batteries from 250 to 700 ampere-hour capacity. The maximum running speed of these locomotives varies from 5 to 18 m.p.h., with a draw-bar pull of 3,900 to 6,000 lb. Smaller locomotives from 2 to 7 tons in weight are also available.

In the German mines the capacity of the battery locomotives varies between 20 and 60 k.w.

A locomotive of German manufacture for main-haulage is shown in Fig. 308, while the British Atlas, 7-ton, main-haulage locomotive is illustrated in Fig. 309, which also shows the battery-changing

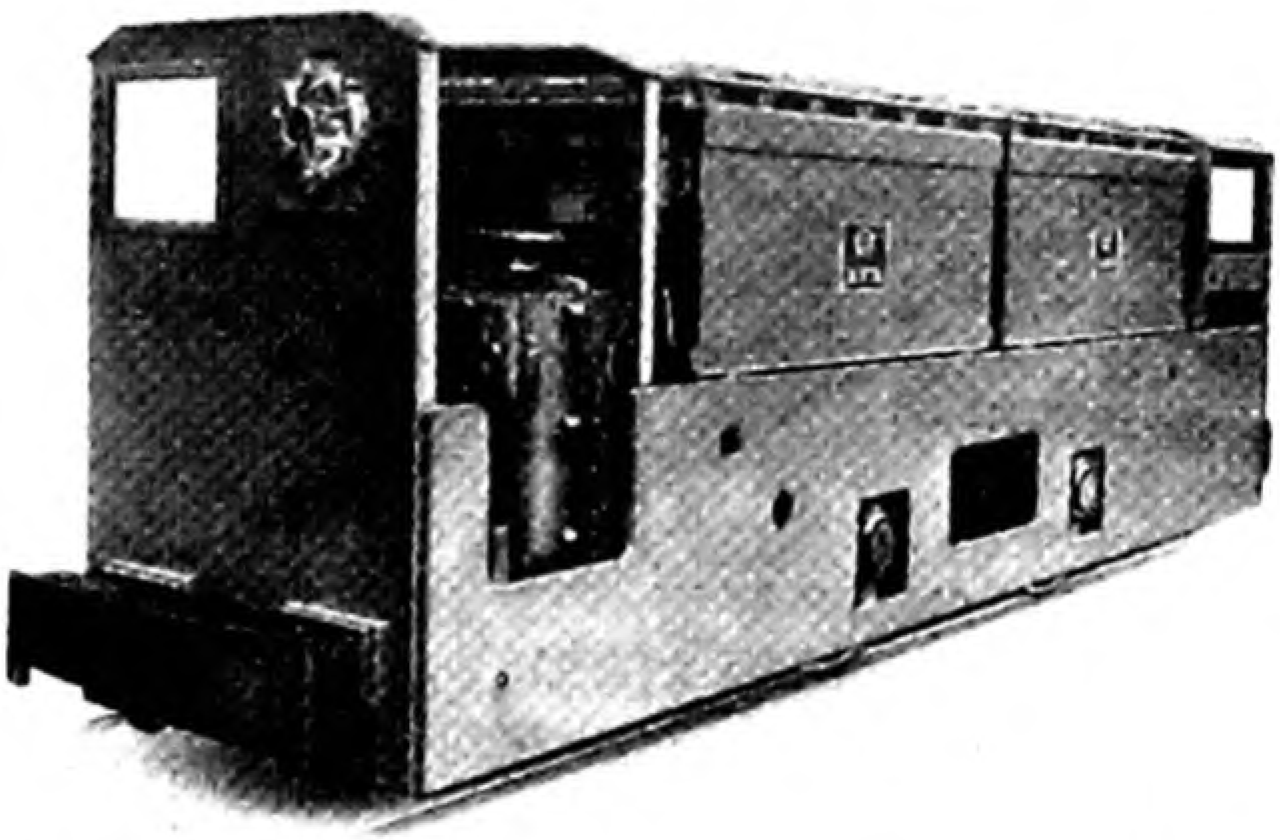
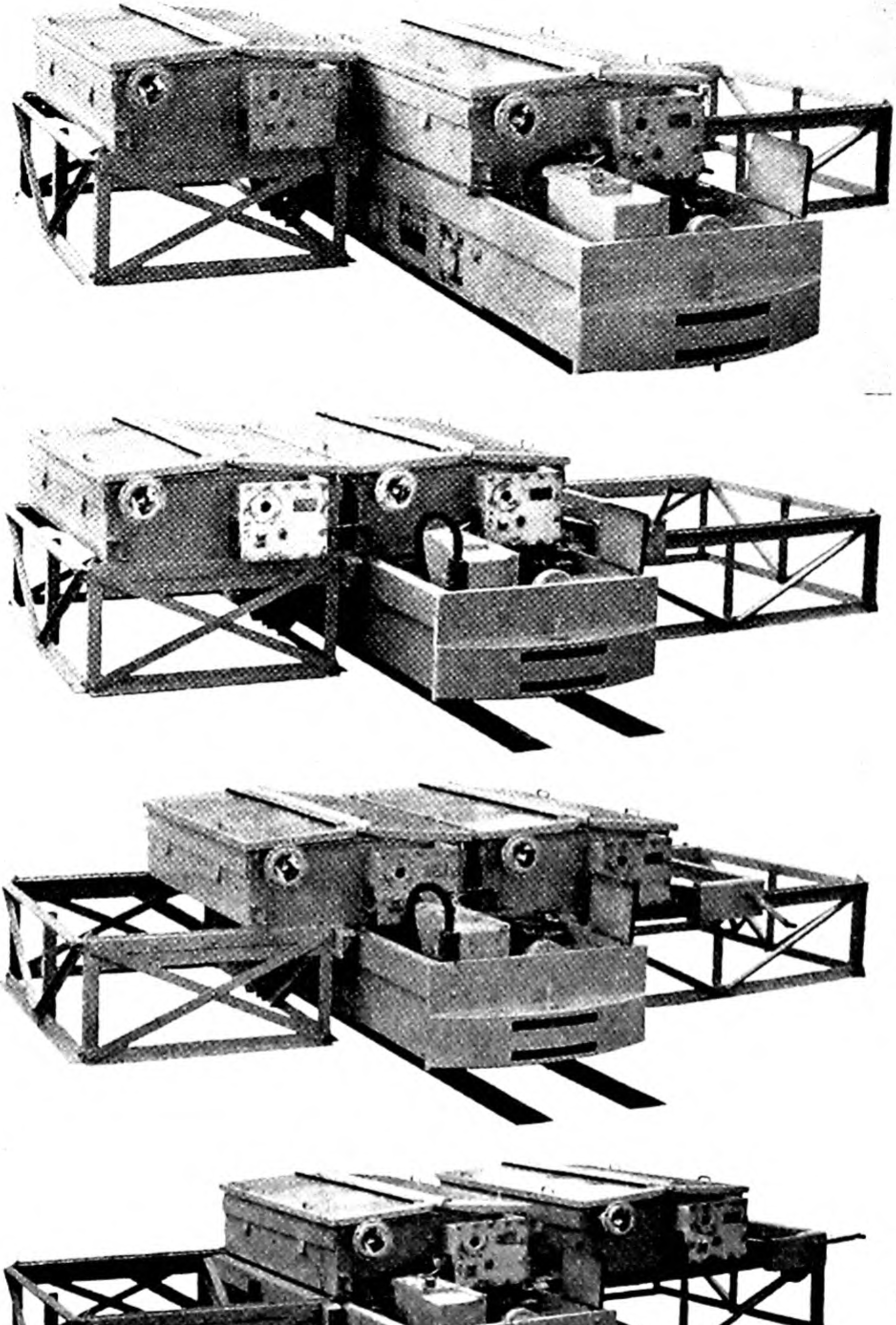


FIG. 308.—Double-engined main-haulage locomotive, 67-h.p., 15 ton.

operation. The Atlas battery locomotive has two motors each of $17\frac{1}{2}$ h.p., one driving each axle. The motors are series wound for operation on 125 volts. Each axle is driven by means of a double-reduction spur-gear drive in a cast-steel housing, the axle gear being mounted between bearings inside the main housing. The controller is of the series-parallel and split-field type, giving an economical system for starting and running at reduced speeds. The controller and resistance units consist of a flame-proof case containing twelve contact-type switch elements, which are operated by the main operating handle, the eight reverser contacts are operated by the reverser handle. The tubular resistance units are flange-mounted on the base of the controller case and run across the end of the locomotive. The length of the locomotive, excluding bumpers, is 11 feet, with four 20-inch-diameter wheels spread over a base of 3 feet 2 inches.



(a)

(b)

(c)

(d)

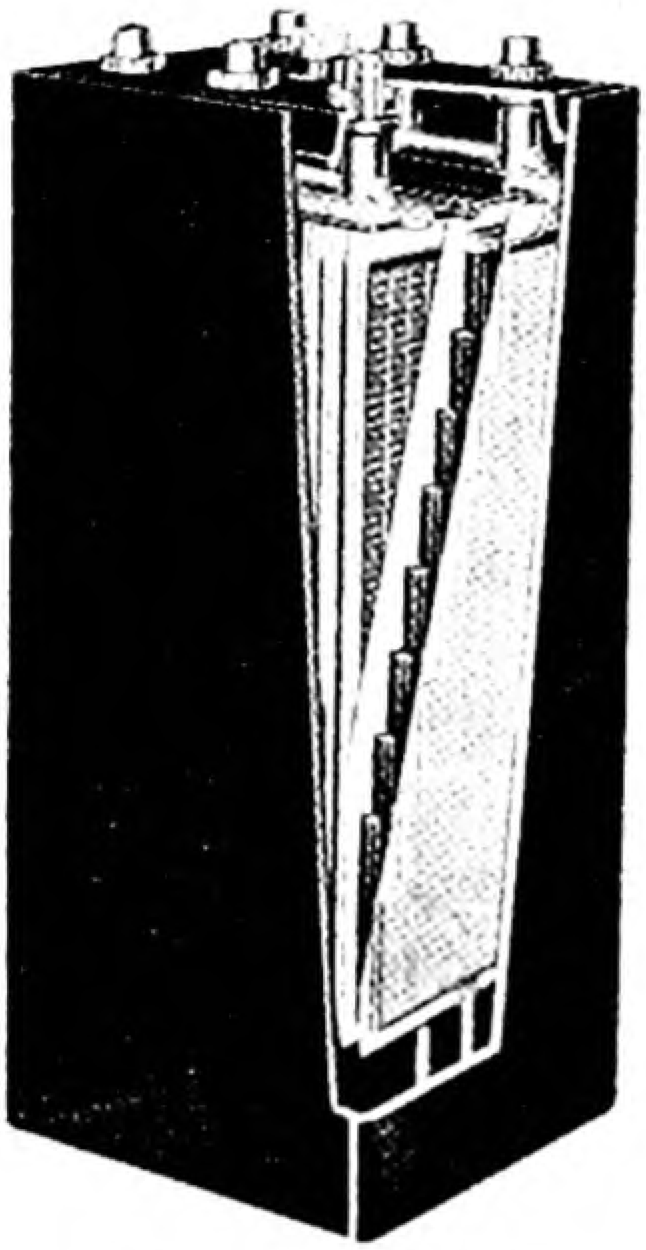
Fig. 309

The width of the locomotive is 4 feet 6 inches, while the height to the top of the battery box is 4 feet. The locomotive can be supplied for gauges from 2 to 3 feet. The rated draw-bar pull is 2,100 lb. at 5 miles per hour, with a maximum of 3,500 lb. The locomotive is arranged for a 66-cell, M.U.V. 15-C., Exide iron-clad battery rated at 253 ampere hours, while larger batteries having a rating up to 374 ampere hours may be fitted if desired.

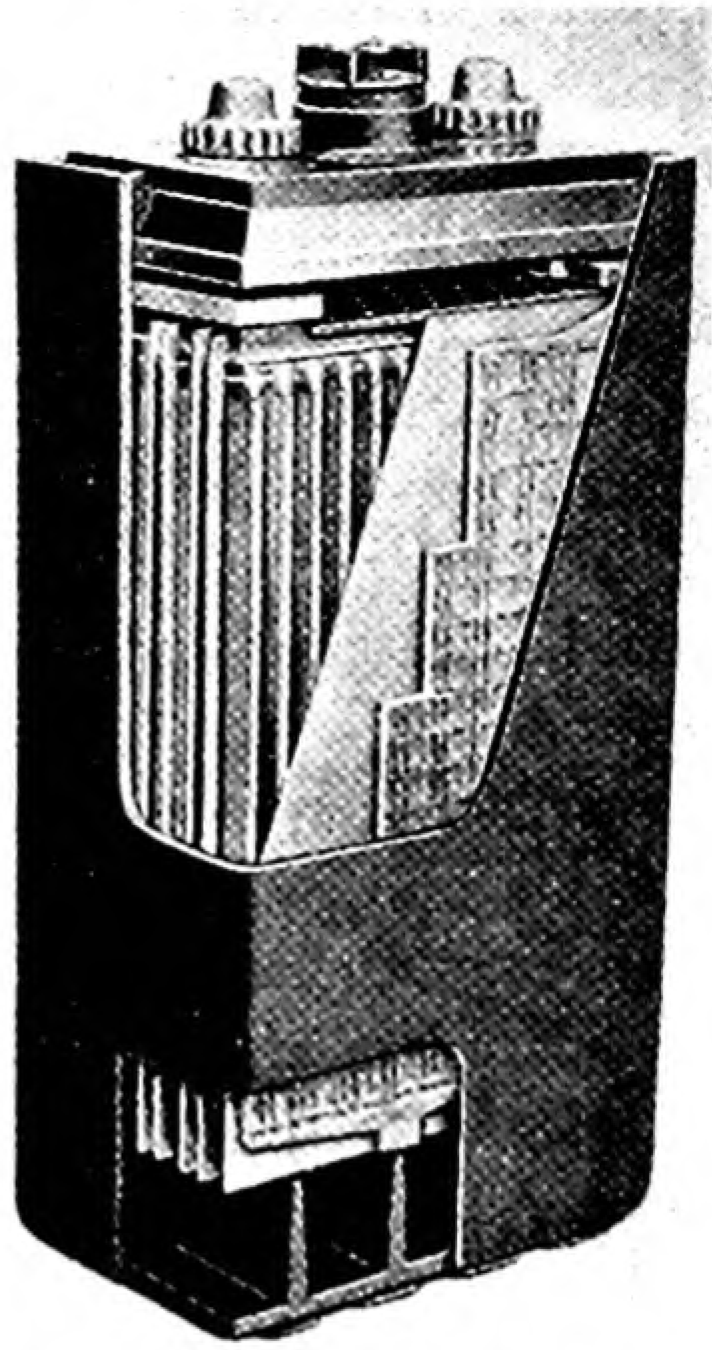
There are various battery designs incorporating cells with armoured plates and lattice-style plates as well as nickel-cadmium cells.

In the armour-plate-type battery, the positive plates consist of a number of ebonite tubes filled with lead dioxide (PbO_2) built up in layers. The tubes are provided with small vertical slits so that the lead dioxide is in contact with the electrolytic acid. In the lattice-plate construction, the grid spaces are filled with the lead dioxide. The negative plates are of lattice construction in both types, the mesh being filled with finely divided lead. The armour-plate battery, shown in Fig. 310, has good durability and is relatively small in size and weight, while the lattice-plate type, illustrated in Fig. 311, has a large capacity with a very low weight and size.

The cost per k.w.h. of storage capacity is twice as great in the armour-plate type as in the lattice-plate type, but on the other hand the latter has to be re-established twice as frequently as the former. Thus, the battery replacement cost per k.w.h. of battery capacity, or per gross ton-mile of operating capacity, balances out for the two battery types. The advantage, however, is gained by the lattice-plate type, which, with equivalent space and weight, can store up to 50 per cent. or more power per charge and thus has a 50 per cent. longer operating time per change. The life is reduced by using the thinner and less durable lattice-plate construction, but the gain in operating capacity more than offsets the lower battery working life. The nickel-cadmium battery, shown in Fig. 312, is less in weight but takes up more space; it has double the life of the armour-plate battery and four times the life of the lattice-plate type. The battery cost, however, is twice as much as the armour-plate type, while both replacement and maintenance costs per k.w.h. are higher. The rate of discharge of the battery in k.w.h. units per gross ton-mile (total load) depends upon the resistance to motion, which can be estimated to be from 8 to 10 kg. per ton gross for subsidiary roads



Exide iron-clad battery.



German armour-plate battery.

FIG. 310

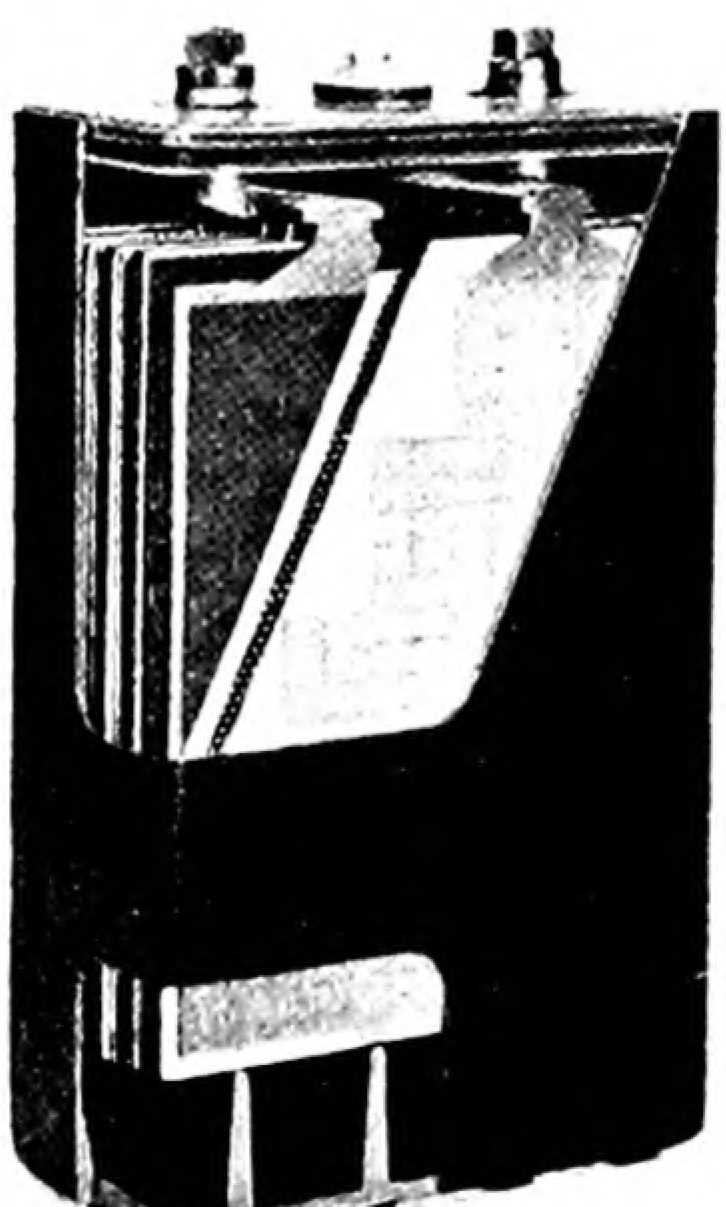


FIG. 311.—Lattice-plate type battery.

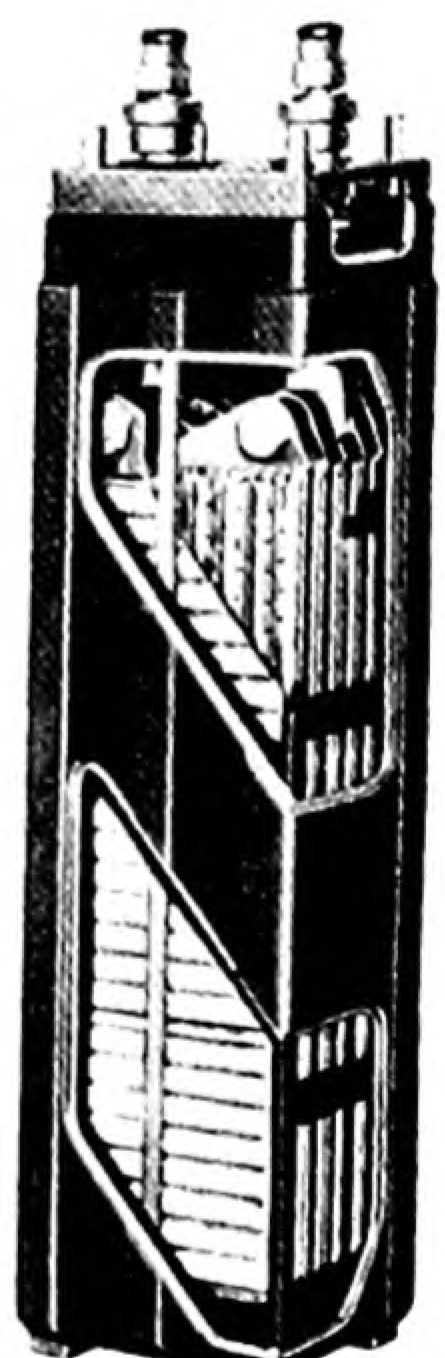


FIG. 312.—Nickel-cadmium battery.

and from 4 to 8 kg. per ton gross for main roads. With a resistance of 6 kg. per ton, a battery performance of 0·04 k.w.h. per gross ton-mile can be assumed. The battery should be operated only up to 85 per cent. of capacity before recharging. In the case of a 40-k.w. main-road locomotive having a battery rating of 98 k.w.h., the locomotive should be operated for not more than $98 \times 0{\cdot}85/0{\cdot}04$, or 2,080, gross ton-miles. When battery locomotives are adopted for main or subsidiary haulage, it is essential to use compact batteries and to maintain a good track and efficient axles and bearings on the mine cars to reduce the friction load to the minimum. Considering all the power losses in the locomotive drive, the power conversion losses with battery operation and the power loss in the changing equipment, the power consumption from the mains can be assumed to be about 0·1 k.w.h. per gross ton-mile.

The average battery voltage adopted in the Ruhr is 72 volts, while the rating for main-road locomotives is between 70 and 200 k.w.h. and for gathering locomotives between 30 and 40 k.w.h. A specially constructed battery casing is adopted in gassy mines, giving flame-proof protection. The cover consists of a series of metal sheets about 1 mm. thick, separated by a gap of 0·5 mm. These sheets are attached to a frame within the battery cover. The depth of this grid depends upon the free space above and between the cells.

Since it is important to effect a quick battery change, the battery is usually put into a container which can be rolled or lifted off the locomotive. Where the battery is rolled off, the locomotive is provided with rollers on the battery bed plate, and a corresponding roller table is provided beside the track at the charging station. Where the rollers on the table and on the locomotive can be coupled together, battery changing can be effected easily and quickly by hand or by a motor-driven device. Where sufficient space is available, the changing station can be provided with tables on each side of the track, so that the replacement battery can be run on to the locomotive as the discharged battery is being removed. The Atlas underground charging-station changing table is shown in Fig. 309, which clearly illustrates its operation. The battery-charging station must be adequately ventilated. The D.C. power required may be generated either with mercury-arc rectifiers in suitable enclosures or by the cheaper dry selenium-cell rectifier

units. The choice of the number of batteries is extremely important, since it will influence the initial and operating costs considerably. The number chosen should be based on utilising the battery capacity from a minimum of from 40 to 50 per cent. up to a maximum of 70 per cent. Since the operation of a battery up to 85 per cent. of its capacity per change is desirable, the number estimated on the basis considered will provide a sufficient reserve to cover the probable unequal operation of the individual locomotives in the mine. The provision of replacement, or change, batteries for each battery in operation can be decided upon only after consideration of the operating conditions.

The flame-proof D.C. motors used are generally suspended within the chassis, the power transmission being through a simple double-stage spur gear. Speed regulation is carried out by using a cam-operated traction switch, which secures regulation without losses. In the case of gate-road locomotives, the traction switch is designed with three steps. The first step couples the halves of the battery in parallel and the motors in series, the voltage across the motor terminals being 18 volts. The second step connects the motors and battery in series, the voltage across the motor terminals being 36 volts. With the final or third step, the battery is connected in series while the motors are coupled in parallel, giving the full voltage of 72 volts across the motor terminals.

For main-road locomotives, the switch is usually provided with six traction steps. The first, third and sixth steps are usually similar to those described for gate-road locomotives. The intermediate steps of the switch bring in a shunt field winding, thus putting in a resistance in parallel with the motor field, reducing the field and increasing the motor speed ratings. The horse-power of battery locomotives are average values, based on an hourly rating and calculated on a basis of the temperature rise in the motor windings. The limiting mechanical performance for a short time is about 100 per cent. higher than the nominal rating, and this additional power can be utilised for speeding up at the start of a run and to cover any temporary overload. The usual drop of about 5 per cent. in the battery voltage, which occurs over a working shift, does not influence the operation of the series-wound motors. In the case of a normal train load, the motors are generally underloaded to such an extent that the train speed is about 20 per cent. higher than the

nominal operating speed. The tractive effort-speed curve for a German battery locomotive is shown in Fig. 313.

(*a*) *Haulage costs with battery locomotives.* Battery locomotives have the disadvantage of high initial cost, which, however, is offset by their long working life of about 20 to 25 years. Including the

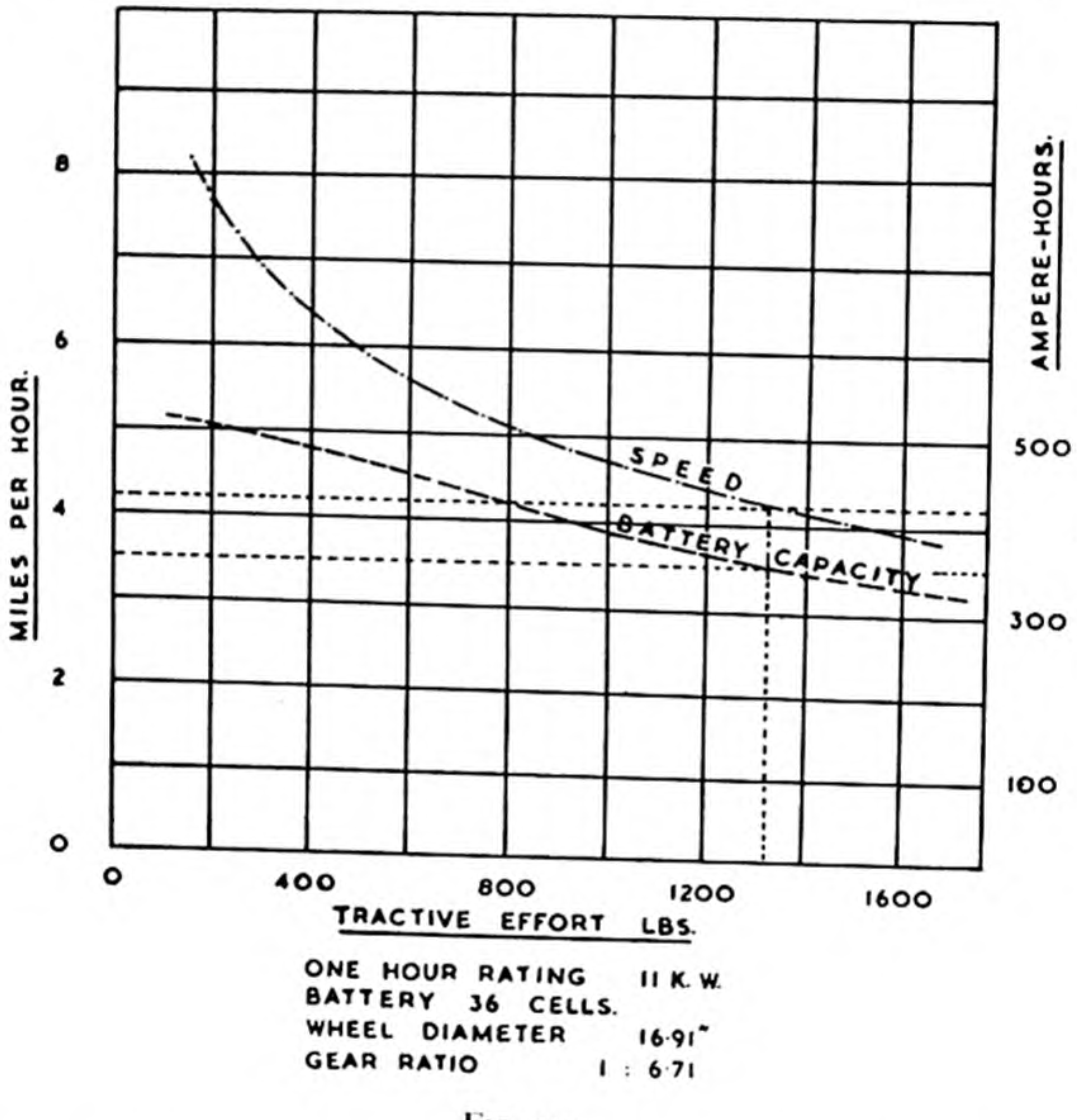


Fig. 313

reserve batteries and the cost of the changing-station installation, the initial capital expenditure can be estimated to be from £150 to £250 per k.w. of locomotive capacity.

The initial cost will depend on the number of locomotives and their full utilisation, together with the cost and proper utilisation of the changing station. Since the number of reserve locomotives will be 15 to 25 per cent. of the operational locomotives, the total initial cost may be expressed as:

$$C = (1{\cdot}25 \text{ to } 1{\cdot}15)\ a \times n \times \text{k.w.}_{\text{Loc}}\ \pounds.$$

Thus, for the operation of 10 locomotives, each of 40 k.w. capacity, the cost would be approximately £100,000.

The annual operational cost will include interest, depreciation, wages, spare parts, battery renewals, lubrication and electric power. The wages cost will include drivers, changing-station attendants and electricians. Where two changing shifts are in operation, 0·4 shift per locomotive per day can be assumed for changing and 0·3 shift per locomotive per day for maintenance and battery cleaning by the electricians, making a total of 0·7 shift per locomotive per day. With less than 5 locomotives, this can be increased to 1 shift, and with a large number of locomotives can be decreased to 0·5 shift. The locomotives require two drivers each, or 2 shifts per locomotive per day. Initially, the cost of spare parts will be small, but will increase with time, the average cost for main-road locomotives being about £80 per locomotive per annum. Battery renewals will cost about £5 per k.w.h. per annum. The consumption of lubricating oil can be estimated to be about 50 k.g. per locomotive per annum. The power consumption from a three-phase supply for battery-changing operations can be estimated to be about 0·1 k.w.h. per gross ton-mile, which value may be decreased to about 0·066 k.w.h. where large mine cars are being used. These estimates all refer to main-road locomotives, and for gathering locomotives the estimates for the cost categories discussed are approximately the same. Since the working performance of the gate-road locomotives is in general only about $\frac{1}{6}$ to $\frac{1}{10}$ of the main-road locomotives, the operational cost per gross ton-mile for the gate-road locomotives will be six to ten times more than for main-road locomotives.

(*b*) *The trolley-wire battery tandem and trolley-wire charging locomotive.* The trolley-wire locomotive has the advantage of a lower operating cost and is always available for immediate use, while the operation of battery locomotives is safer in gassy conditions. The possibility of taking advantage of the characteristics of both trolley-wire and battery operation has led to the development and introduction in some mines in the Ruhr of two types of combined trolley-battery locomotives, the trolley-wire battery tandem locomotive and the trolley-wire charging locomotive. The fundamental feature of these developments is to use the overhead line over the longest possible distance and to switch over to battery operation only in

those sections of the run where the methane content of the air in the haulage road exceeds 0·3 per cent.

The trolley-wire battery tandem locomotive shown in Fig. 314 carries a battery trailer with it for use when the trolley-wire operation is suspended. The battery is charged at the charging station in the normal manner.

The trolley-wire charging locomotive charges the battery itself from the overhead power line.

Neither of these locomotives is completely satisfactory, since in

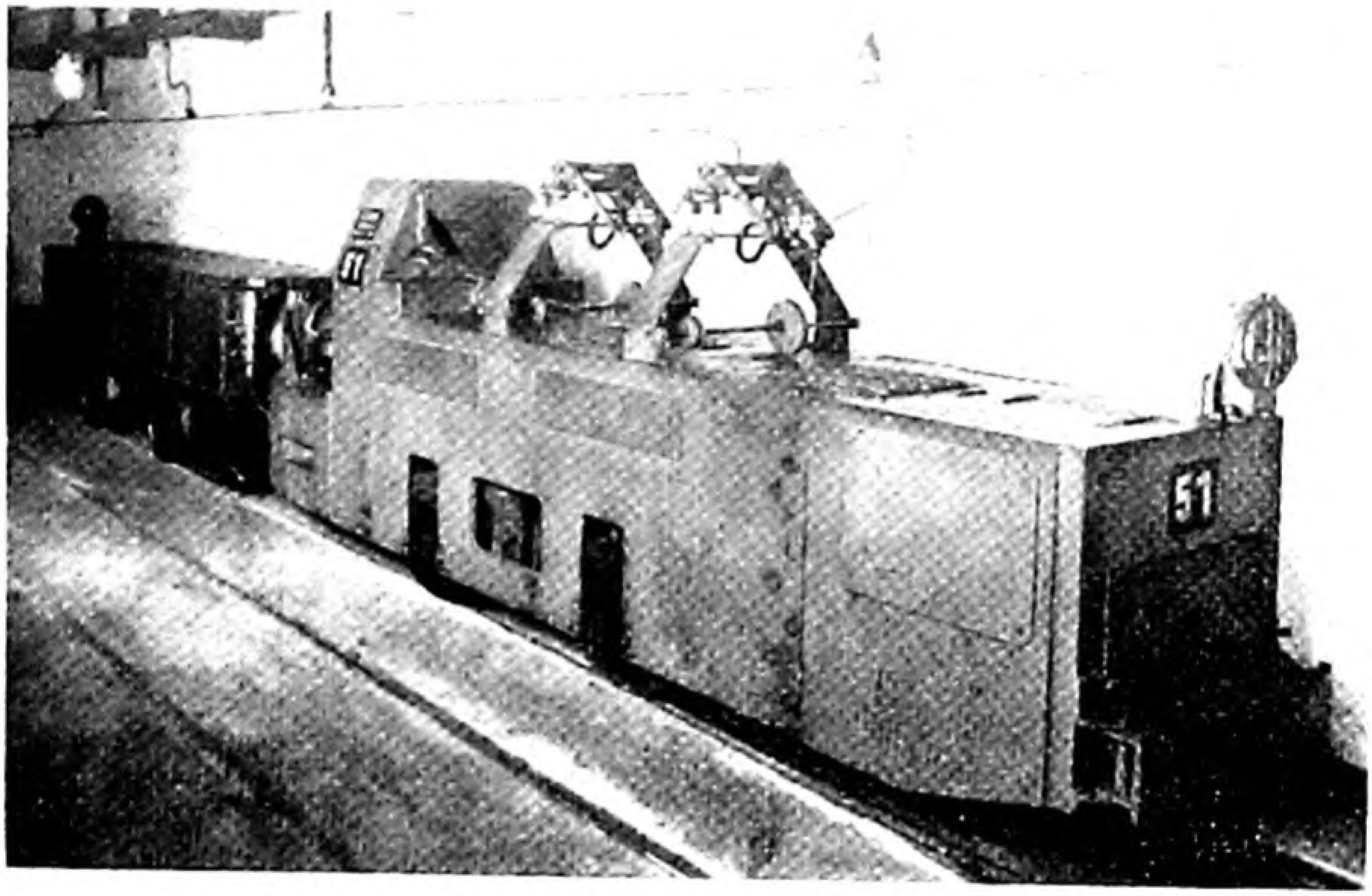


FIG. 314

addition to the increase in the first cost, the operating efficiency is affected by the additional load of the battery carried. Experience has shown that the battery of the charging locomotive must be given a full charge every two days at the charging station and there is, however, the possible danger of an over-charge due to the failure of the automatic cut-off device, producing an inflammable oxyhydrogen gas. The application of these locomotives has been limited for the reasons stated and it is usual to find gathering locomotives, of either the diesel or battery type, used on sections where it is not considered desirable to run trolley-wire locomotives.

Section 3. Trolley-wire Locomotives

Introduction. The trolley-wire locomotive, which is used widely in American and European coal mines, is an electric locomotive fed from an overhead conductor, and has achieved its popularity for a variety of reasons. The main factor is probably the higher power/space ratio which can be obtained with the trolley-wire type in comparison with other locomotives for underground use. The tractive effort, and therefore the draw-bar pull, developed by any locomotive is essentially dependent upon its own weight, while the speed of operation is governed by the horse-power of the prime mover which has been built into the locomotive chassis. There have been no trolley-wire locomotive installations in British coal mines, since their use is excluded under G.R. 136 (*a*) subject, however, to exemption under G.R. 137 (*a*).

The following table * shows the approximate numbers of various types of locomotives in use in the Ruhr coalfield on 1st January, 1940 and in December 1948.

Type	*Number in Use*	
	1st Jan. 1940	*Dec.* 1948
Trolley-wire	1,270	1,202
Trolley/Battery	24	35
Battery	358	437
Compressed air	1,223	1,307
Diesel	428	564
Benzol and others	8	22

The changes in the numbers given are not comparative, since these appear to have been largely determined by the relative availability of materials. It should be noted that main-road and gate-road locomotives are included in the numbers given.

The use of trolley-wire locomotives in certain British mines is almost certain to occur, but it is probable that their use will be limited to main haulage where the ventilation conditions are good, and full advantage can be taken of the essential features of this type, in which increases in weight and horse-power can be made without appreciable increase in overall dimensions. It is also necessary to

* N.C.B. Information Bulletin No. E.G. (50) 1.

have a large capacity sub-station from which the locomotive can draw indefinitely.

The design of the trolley-wire locomotive is simple and robust, embodying few moving parts, so that maintenance requirements are readily covered and the locomotive is capable of reliably standing up to the heavy work typical of underground conditions.

As the use of trolley-wire locomotives is limited in Britain by reason of the safety factors involved, these factors are discussed in the following section.

(*a*) *Possibilities of danger connected with trolley-wire locomotives.* The use of trolley-wire locomotives in the Ruhr is limited to roadways in which the methane content is less than 0·3 per cent. The possibility of a fire-damp explosion where trolley-wire locomotives are in use can only be met effectively by restricting their use to intake air roadways and providing adequate ventilation. The constant supervision of the methane content of the air is essential, and in the Ruhr, automatic recorders for giving the velocity, quantity and methane content of the air are in course of development. Such instruments will be installed at appropriate points within the main haulage roads and will automatically operate an alarm signal or disconnect the overhead line from the supply in the event of conditions arising when the fixed limits laid down are exceeded in any particular road section. Local accumulations of methane can be diluted by providing adequate ventilation. To assist to this end, the roof and sides of roadways which have not been brick-lined or are standing unsupported should be well lagged, all cavities should be filled up and intersected seams or faults should be covered by compact lagging.

A more important precautionary measure is the reduction of arcing at the collectors on the overhead line. The ignition of gas is most likely to be caused by arcing at the collectors, and the possibility of direct coal-dust ignition should not be excluded. The track should be well laid to guarantee smooth running of the locomotive, while the overhead-line suspension should be flexible, have little sag and be maintained as nearly parallel to the rails as possible. The U.S. Bureau of Mines recommends that the sag should not exceed 1·5 inches on a 30-foot span and that the gradient between overhead wire and rail should not be more than 2 per cent. In the Ruhr, the insulator spacing recommended is from 15 to 20 feet on straight

roads and from 10 to 12 feet on curves, which precludes any noticeable sag.

In order to maintain an even gradient between the overhead wire and the rails, the suspension device should be able to accommodate changes in the floor- or roof-level due to mining conditions. The locomotives should be provided with two independent collectors, so that at least one collector will be in contact with the overhead line and so avoid arcing. The overhead line and contact area of the collector should have a smooth polished surface, which can be obtained by slight lubrication of the wire and/or by using carbon collecting shoes. The importance of keeping a minimum distance between the roof or supports and the overhead wire is stressed in trolley-wire locomotive practice in the Ruhr, where the limitation of 30 cm. (about 12 inches) is prescribed by regulation. This restriction is laid down to keep possible arcing at the collectors at a reasonable distance from any methane accumulation near the roof and the collectors. Track-limit switches are generally used to disconnect the last 30 feet of the overhead line from the supply, should the locomotive overrun the line. These switches are installed to prevent the formation of sparks at the collectors, which would occur in the event of an overrun with the motor running. In some instances the switches may be removed and the trolley-wire replaced with a wooden guide over the last few yards of track.

It is probable that in British mines, where the haulage roadways are generally high enough to give adequate clearance for the locomotive, the overhead wire will be placed centrally with the track. In this case, the pantograph type of collector, as illustrated in Fig. 315, will be the probable choice. With this type of collector it is claimed that less arcing occurs, while it has the advantage of being suitable for either direction of travel without alteration. With the pantograph system, it is impossible to protect the overhead wire with side-boards at points where men entrain or where precautions against the accidental contact with the overhead line are desirable. In the case of a central overhead wire system, there is the possibility of a broken wire falling on the train, and it is therefore necessary to provide cab protection of a proper design on the locomotive and on the man-riding train if this is to be used. In the Ruhr, it is stated that the danger of contact with the overhead line is minimised by efficient traffic control, as in other industrial traffic operations. The

minimum clearance between rail and overhead line prescribed is 1·8 m. (about 6 feet) where the voltage is not more than 220, and 2·2 m. (about 7·25 feet) where the voltage is between 220 and 550. The overhead line is disconnected during changes of shifts or during man-haulage by other means. It appears that over many years these measures have been sufficient to limit accidents by contact with the overhead line to a very low rate.

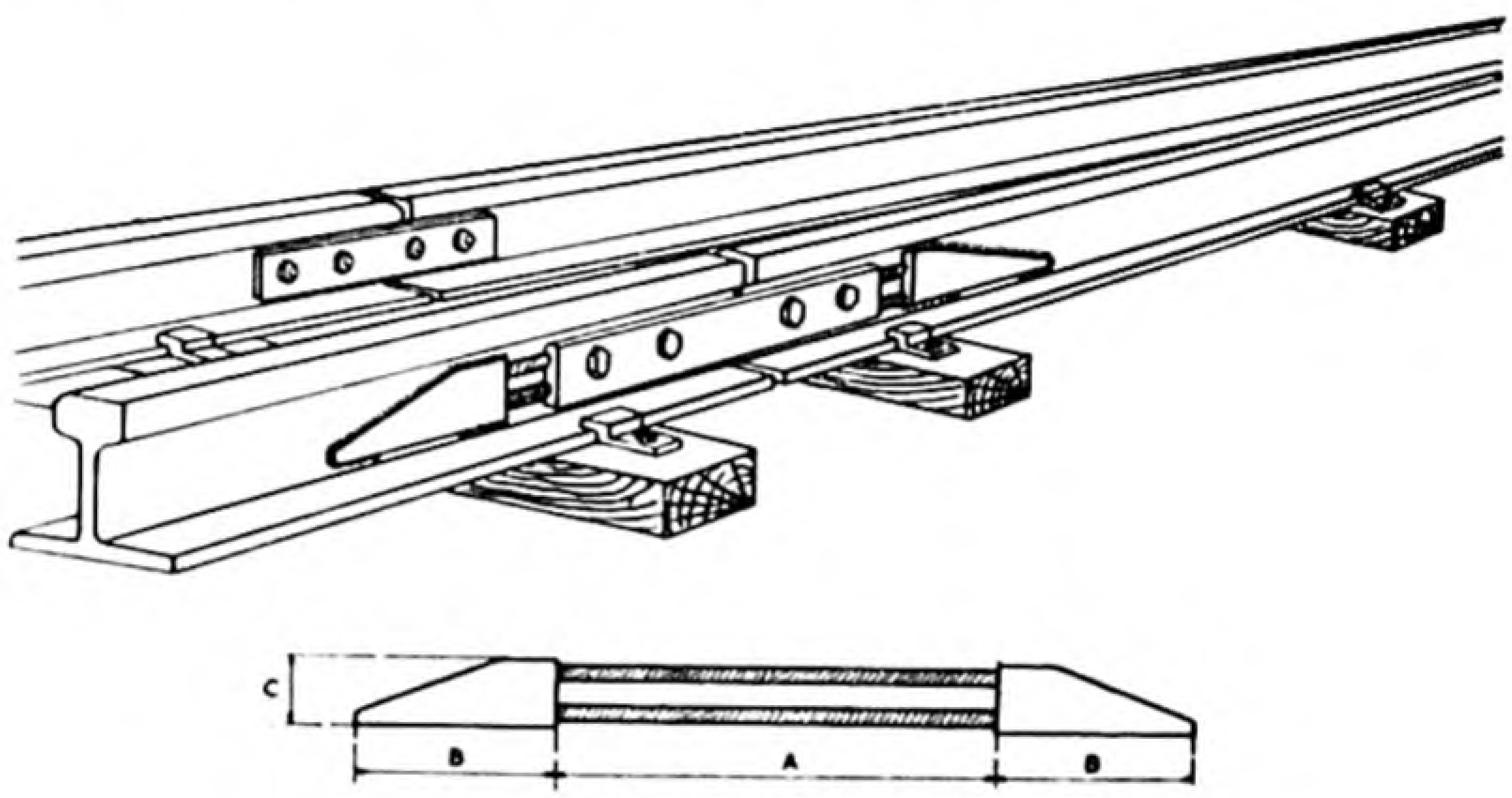
Current leakage is a further danger which may cause serious accident if the current is conducted into a working section of the mine via pipe-lines, air-ducts, ropes, etc. In this case, high-tension detonators will be required for shot firing. A considerable reduction of this danger is achieved if an efficient rail-earth return system is

Fig. 315

installed. In a track-return system, earth-leakage protection cannot be provided to limit earth-fault current, as this cannot be distinguished from load current. The tendency in modern practice is towards welded rail joints and it is probable that this type of joint will be preferred in British practice. Where it is not possible to use a welded joint because of floor movement, the form illustrated in Fig. 316 * may be used, in which the rails are joined mechanically by standard four-bolt fish-plates and electrically by copper bonds. The copper bonds, as shown in Fig. 316, consist of stranded conductors, each of 25-sq. mm. (about $\frac{1}{25}$ sq. in.) cross-section, brazed to iron ends which are welded to the rail webs. The stranded conductors are protected by the fish-plate, the latter being grooved to accommodate them.

* N.C.B. Information Bulletin No. E.G. (50) 1.

The danger from fire caused by a short circuit of the system is also a hazard which should be noted. This occurrence, however, exists only in the case of a prolonged leakage over wood, such as timber supports, and can be effectively prevented by an efficient layout of the electrical system, incorporating protective switchgear. In mines in which a trolley-wire system is installed, it is imperative that electrical systems should be sectionalised, so that adequate protective switchgear may be available close to the leakage in the event of a dead short-circuit in the system. The usual operating



TYPE	A	B	C
L 700	700 M/M (27·6")	120 M/M (4·7")	42 M/M (1·7")
L 500	500 M/M (19·8")	120 M/M (4·7")	42 M/M (1·7")

FIG. 316

voltage in the Ruhr is 220, while in American coal mines 500 is the usual voltage. The choice of an operating voltage of either 220 or 500 has little effect on the risks involved due to gas or coal-dust ignitions. There is only a slight possibility of a fatal shock from a 220-volt supply, which would be the case with a 500-volt system. The risk of accidental contact has already been discussed and it would appear that suitable precautions can be effected and the danger minimised.

(*b*) *Choice of current supply.* In the Ruhr, D.C. at 220 volts is normally used, though there are one or two A.C. installations, but further A.C. schemes have been prohibited since 1910. The main deciding factor was the lesser danger of body contact with D.C. The draft British regulations also preclude the use of A.C. supply. The torque-speed characteristics of D.C. series-wound motors are

ideal for traction work, while the control of the motor is simple. The possible use of single-phase A.C. power has been considered. With A.C. series-wound single-phase motors, the torque-speed characteristics are similar to those of the D.C. motor, but the former are larger and more expensive. With a universal single-phase motor, the same control gear can be used as for D.C., and the extra cost of the motors is offset by the cheaper distribution of A.C. In German

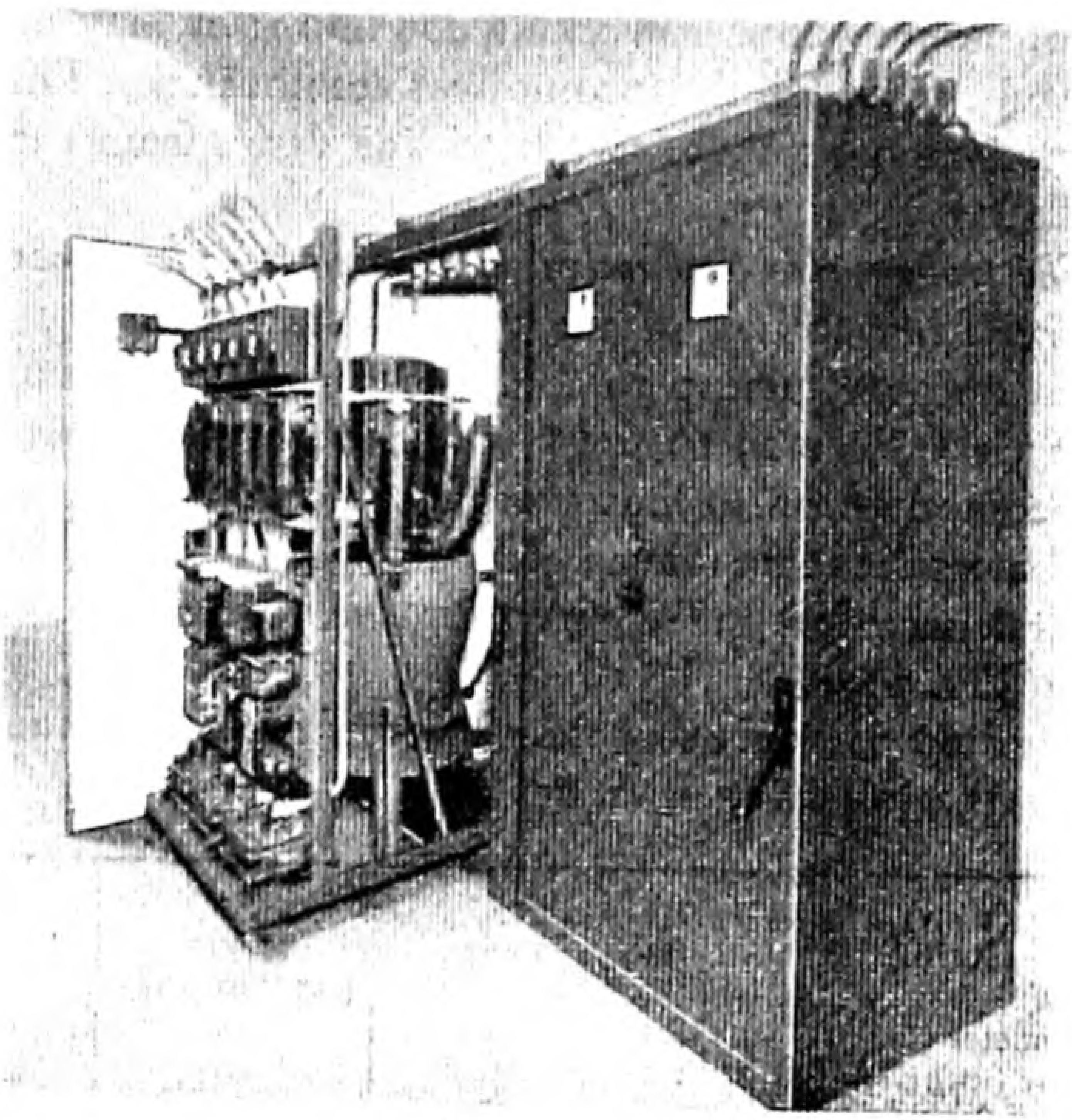


Fig. 317

practice, in order to reduce the inherent voltage drop of a D.C. system, the D.C. is converted from A.C. by converting units installed near the shaft. These units are rotary converters or mercury-arc rectifiers with the necessary switch gear and transformers. The incoming supply is usually 5,000 volts, 3-phase, 50 cycles. The rotary converters, which are still widely used, will probably be replaced eventually by mercury-arc rectifiers of the glass-bulb or steel-tank type, the latter being favoured due to its longer life. The two types are illustrated in Figs. 317 and 318. Rectifiers of these types are constructed for capacities of 400 and 600 amps. at 220 volts.

In order to compute the required capacity and the number of units, the maximum load under normal conditions has to be determined, taking into consideration that all the locomotives are not working under full-load conditions simultaneously. The normal maximum load is taken as from 0·7 to 0·8 of the installed locomotive capacity, of which from 25 to 35 per cent. of the installed electric capacity is included as a reserve. In the Ruhr, an average overhead-line load of 53 k.w. per mile of line is assumed, which may increase to 80 k.w. per mile at peak working conditions or may drop to 30 k.w. per mile where all the locomotives are not in use. The number of locomotives per mile of overhead line varies from 1 to 2·2, the average being about 1·5.

(c) *The trolley-wire locomotive with overhead-line suspension and rail return.* Locomotives of the type illustrated in Fig. 315, for operation on gradients up to 1 in 200, are generally constructed with capacities from 25 to 100 k.w. for use in the Ruhr. In many cases, locomotives of 40 k.w. will be sufficient, while units of 70 k.w. will meet the largest demands.

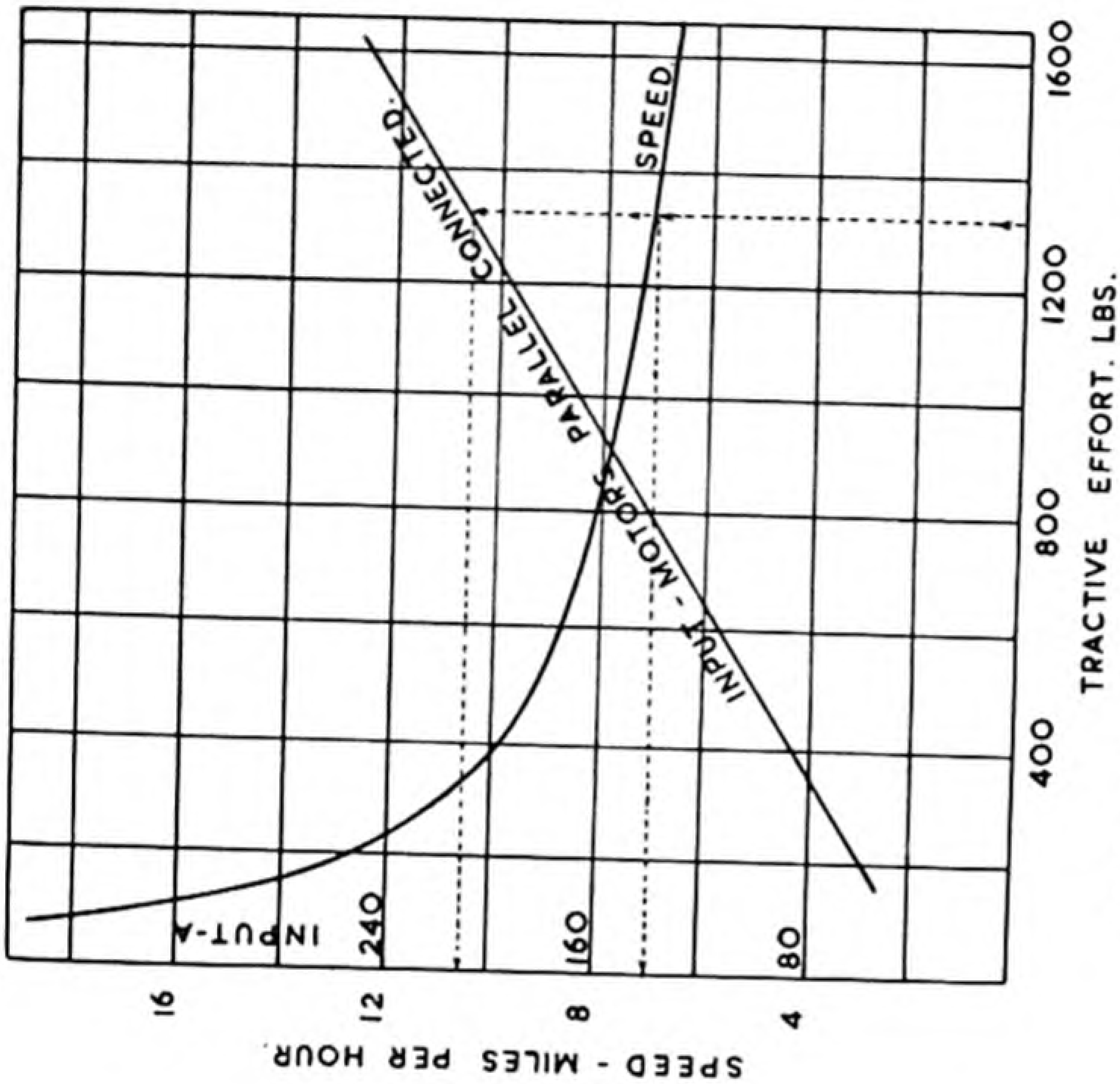
The tractive effort-speed curve for the Siemens locomotive is shown in Fig. 319.

The leading particulars of three sizes of trolley locomotives are shown in the following table *:

H.P. rating (1 hr.)	52	85	100
Draw-bar pull (lb.)	2,460	3,960	4,310
Gauge (in.)		21 to 25	
Wheel diameter (in.)	31	31	31·9
Wheel base (in.)	43	43	50
Weight (tons)	8	13	12
Length overall	15 ft. 11 in.	16 ft. 3 in.	15 ft. 10 in.
Height overall	5 ft. 3 in.	5 ft. 3 in.	5 ft. 3 in.
Width overall	3 ft. ½ in.	3 ft. 3½ in.	2 ft. 11½ in.
Working voltage	220	220	220
Radius of smallest curve	26 ft. 3 in.	26 ft. 3 in.	30 ft.

Locomotives of greater horse-power rating will be considered only where heavier gradients have to be negotiated. In cases where the roads have slight gradients or where mine cars of up to only 1 ton capacity are used, larger locomotives can be justified only if long trains are hauled. The haulage of long trains will be dependent upon the siding accommodation available for the mine cars near the

* N.C.B. Bulletin No. E.G. (50) 1.



ONE HOUR RATING - 40 K.W. AT 220V. 2 MOTORS 20 K.W.
WHEEL DIA. 30·5"
GEAR RATIO 1: 7·92

FIG. 319.—Tractive effort speed-curve for Siemens trolley locomotive EL.6.

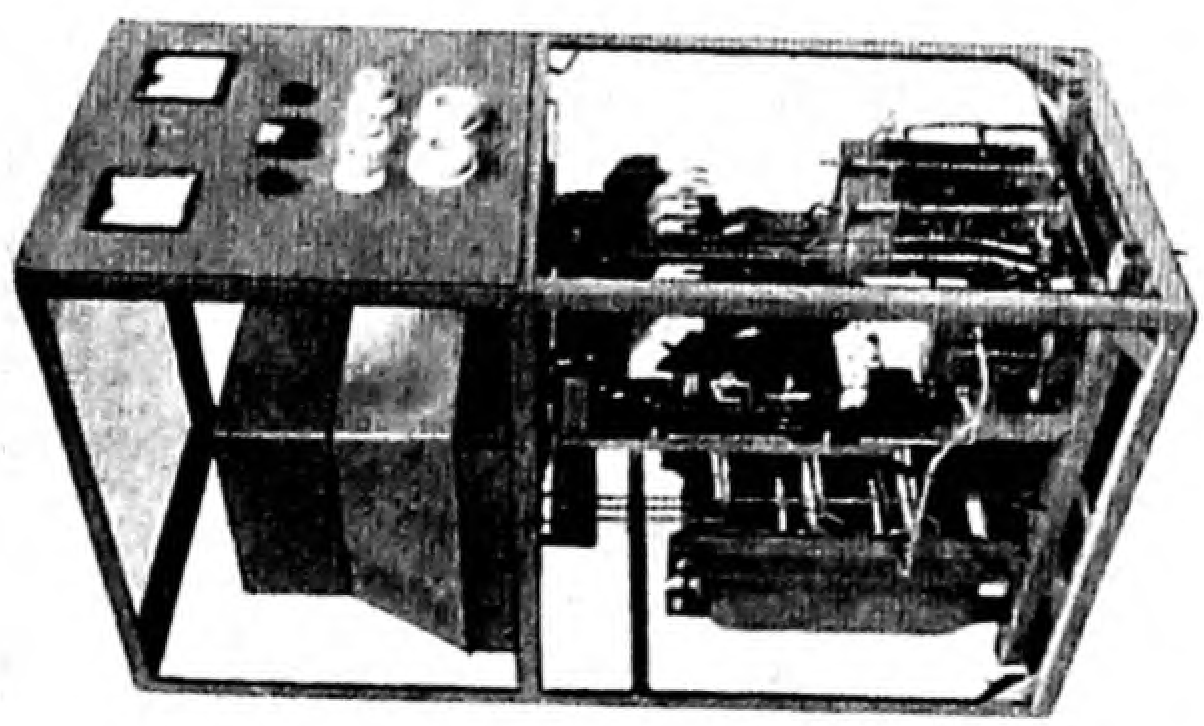


FIG. 318

staple shafts and the main loading stations. Where larger mine cars of, say, from 3- to 5-ton capacity are intended to be used, shorter trains can be assembled and hauled. The braking of such trains presents an additional problem, since locomotive braking alone is insufficient.

The locomotives used in the Ruhr generally adopt two direct-current series-wound motors running in roller bearings, each one driving an axle through case-hardened, straight-cut, single-reduction toothed gearing. The traction motors are electrically coupled for series-parallel control. The traction switch has generally nine steps. Braking is performed by hand-operated brake blocks acting on the wheel treads and by short-circuiting the motor so that it acts as a generator and dissipates the energy through resistances. Counter-current braking is not recommended, since this method would heavily overload the motors. Both air-break and oil-immersed short-circuit switches are used, the provision of such switches being compulsory. The traction control, hand-brake, sand control, main switch and short-circuit switch are installed in the roofed cab. With the modern cam-operated traction switches, the main switch and short-circuit switch are incorporated into the traction switch, thus saving a great deal of space. The cam-operated switch can also be provided with a no-volt release which acts in case of a failure of the supply, releasing the main switch and retaining it in the inoperative position in the event of a restoration of the supply, until reset by the driver.

The collectors previously referred to should be constructed in such a manner as to minimise arcing as far as possible. The pantograph collector or bow collector are recommended, since the spring loading incorporated ensures positive combat on the overhead line. The collector shoes usually used in the Ruhr consist of hard carbon blocks embedded in copper rod. The type which incorporates an aluminium shoe and felt oil pads is not recommended.

The overhead line is usually hard-drawn copper wire of grooved section, the normal section being 0·124 square inch. In the case of larger haulage distances, the wire section may be increased to 0·18 or 0·23 square inch in order to keep the voltage drop in the line to a minimum without the necessity for an additional feeder cable for the overhead line.

The overhead-line suspension system recommended so that

arcing is reduced is by transverse wires spanning the roadway. The insulators are held by these transverse wires, which are insulated at the roadway sides. It is claimed that the maximum flexibility is

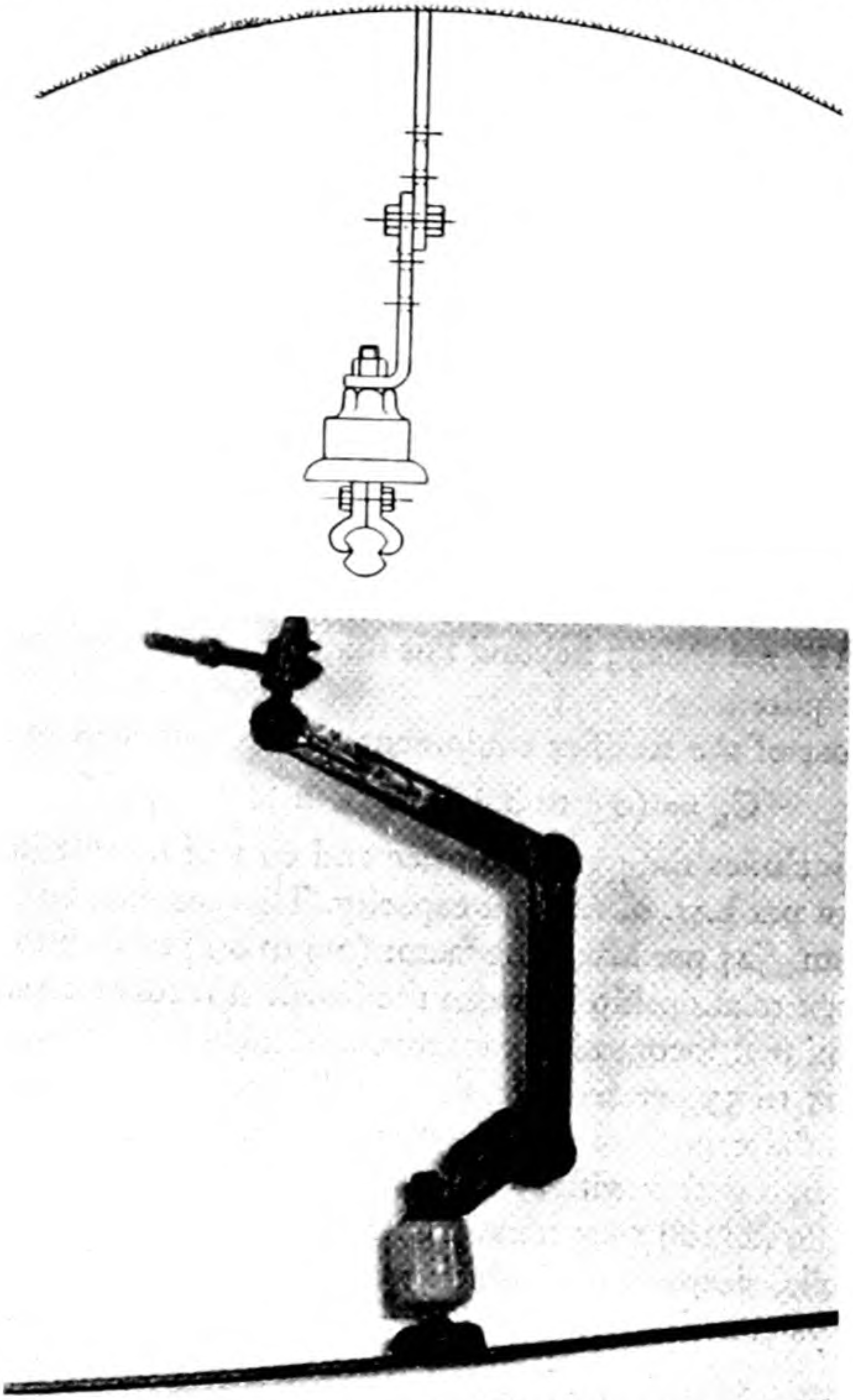


FIG. 320

obtained with this method of suspension. Several forms of insulators have been devised which allow a certain flexibility in movement and which can be adjusted in height to compensate for any roof movements. Two examples of these insulators are shown in Fig. 320.

(*d*) *Haulage costs with trolley-wire locomotives.* The initial capital cost for trolley-wire locomotive haulage includes the cost of the locomotive, the rectifier equipment and the installation of the overhead line.

The initial capital lost for locomotives may be expressed as:

$$C_{Loc} = 1{\cdot}3 \times n \times p \times \text{k.w.}_{Loc}\ £,$$

where n is the number of operational locomotives and p the capital cost per locomotive k.w. The latter will depend upon the locomotive capacity and decreases as the capacity increases.

In Germany, the capital cost per locomotive k.w. for locomotives of 38 k.w. is about £100, and for a 64-k.w. locomotive about £80. The reserve factor 1·3 implies a 20 per cent. locomotive reserve in addition to operational locomotives, and also includes the provision of a sufficient stock of spare parts for all locomotives. With 15 operational locomotives, 2 reserve locomotives will be sufficient, while for from 15 to 25 operational locomotives, 3 in reserve will be necessary; beyond this number, 4 reserve locomotives will be required.

The cost of the rectifier equipment can be estimated as follows:

$$C_R = (0{\cdot}9 \text{ to } 0{\cdot}8) \times b \times n \times \text{k.w.}_{Loc}\ £,$$

where b includes the purchase price and cost of installation of the equipment per k.w. of rectifier capacity. This cost can be estimated to be about £25 per k.w. The factor (0·9 to 0·8) takes into account the average relationship between the installed rectifier capacity and the operational locomotive capacity, and includes a reserve factor of from 25 to 33 per cent. on the installed rectifier capacity.

The initial capital cost of the overhead line includes the cost of installation, together with all accessories, including the provision of an adequate rail return system. The cost does not include the initial cost of rails, sleepers, etc., which would be required for any track-haulage system. The cost for the overhead line can be estimated as follows:

$$C_t = t \times l,$$

where t is the initial capital cost per mile of overhead line and l is the length of the line in miles. The former may be estimated to be £900. On the average, l is equal to 1·6 times the length of haul, which implies that the greater part of the haulage has a double overhead line and the remainder a single overhead line.

Combining the three expressions discussed, the following equation expresses the total initial capital cost of the trolley-wire locomotive installation.

$$C = \{[1\cdot3\ np\ \text{k.w.}_{\text{Loc}}] + [(0\cdot9 \text{ to } 0\cdot8)\ bn\ \text{k.w.}_{\text{Loc}}] + [tl]\}\ \pounds.$$

Where locomotives of 40-k.w. capacity are installed and 1·6 locomotives per mile are in use, the initial capital cost can be estimated to be about £200 per locomotive k.w.

The operational cost will include costs for the supervision and maintenance of the locomotives, rectifier equipment and the overhead-line installation as well as the operation and depreciation costs.

The locomotives maintenance cost will depend on the age of the locomotives and their condition of service. Initially the repair cost will be small, but at a later stage the spare parts cost may be taken as £3 per operational locomotive k.w. per annum and maintenance as 1·3 manshifts per operational locomotive k.w. per day.

The spare parts cost for the rectifier equipment is very small and can be estimated to be from 2s. to 6s. per operational locomotive k.w. per annum. The wages cost for supervision can be taken as 1 shift per day.

The maintenance cost of the overhead line and rail return will include a wages cost and materials cost, and can be estimated to be between £150 and £250 per mile per annum. The locomotive operational cost includes the wages cost, lubrication cost and power cost. The computation of the wages cost can be based upon an estimate of 2·1 manshifts per operational locomotive per day. The power consumption, including the power used for lighting and welding work, can be taken as 0·14 k.w. per gross ton mile. This consumption is measured on the A.C. input side to the rectifier, and thus includes rectifier losses but not transmission losses from the surface to the rectifier sub-station.

The depreciation costs must be computed on the basis of the life of the individual sections of the whole installation. In general, the life of the overhead line is the same as the life of the haulage road. The life of the rectifier equipment and the locomotives can be taken as twenty-five years.

Section 4. Compressed-air Locomotives

The compressed-air locomotive is widely used in the Ruhr and in Holland, but has never been used in coal mines in Britain. The

construction is simple, consisting of motor and driving gear. The compressed air is stored in one or several large cylindrical containers at a pressure of about 2,500 to 3,500 lb. per square inch and passes into a smaller intermediate cylinder, or 'working bottle', before being fed to the motor. As in the case of steam locomotives, the motors are generally of the horizontal-piston type with high- and low-pressure cylinders, while in some cases an intermediate pressure cylinder is introduced. The highly compressed air from the containers passes through a reducing valve and then to the engine at a pressure of from 350 to 430 lb. per square inch for the three-stage expansion engine and from 210 to 260 lb. per square inch for the two-stage expansion engine. The loss of heat due to the expansion of the air may be from 25 to 30 per cent. of its initial energy, but much of this can be returned to the air by passing it through a heater placed before the high-pressure cylinder in the two-stage engine and before the low-pressure cylinder in the three-stage engine. The heater is located within air-inlet tubes through which comparatively warm air from the roadway is drawn. The fact that so great an expansion is possible in the motor cylinder is due to the dryness of the compressed air. In spite of the high dryness factor, water does accumulate within the high-pressure reserve containers and the working cylinder. This water must be removed from time to time in order to prevent it from entering the cylinders, where it may cause water hammer.

When starting up, it is necessary to let air from the working cylinder into the low-pressure cylinder, and for this the control mechanism is fully opened. When stopping the locomotive, the starting valve is closed and the control mechanism put back to the closed position. A 40-h.p. compressed-air locomotive is shown in Fig. 321. The driving wheels are normally located outside the body of the locomotive in order to secure good supervision and easy maintenance.

Four- or six-cylinder radial motors may be used instead of the horizontal-piston type. These motors run at about 1,300 r.p.m. and in certain cases up to 2,000 r.p.m. The compressed air, at a pressure of from 375 to 450 lb. per square inch, is distributed to all the cylinders through a ring main. The air enters each cylinder through an inlet valve and at the end of the piston stroke, when the piston is near the outer dead-centre position, leaves the cylinder through exhaust slots. The air remaining in the cylinder is compressed during

the compression stroke, thus heating itself and the cylinder and so preventing the freezing of the cylinder during the subsequent expansion stroke. The cylinders all act on a common crank shaft, the twisting moment being transmitted to both axles by means of a bevel gear and roller chains. This form of drive has the advantage of demanding less space, and allows the maximum accommodation for the air-reserve containers. The outlet from the air-reserve containers and the direction of rotation are controlled through a hand-lever which operates a disc cam to which are attached five control levers. These levers, actuated from the crank pin, operate the valve

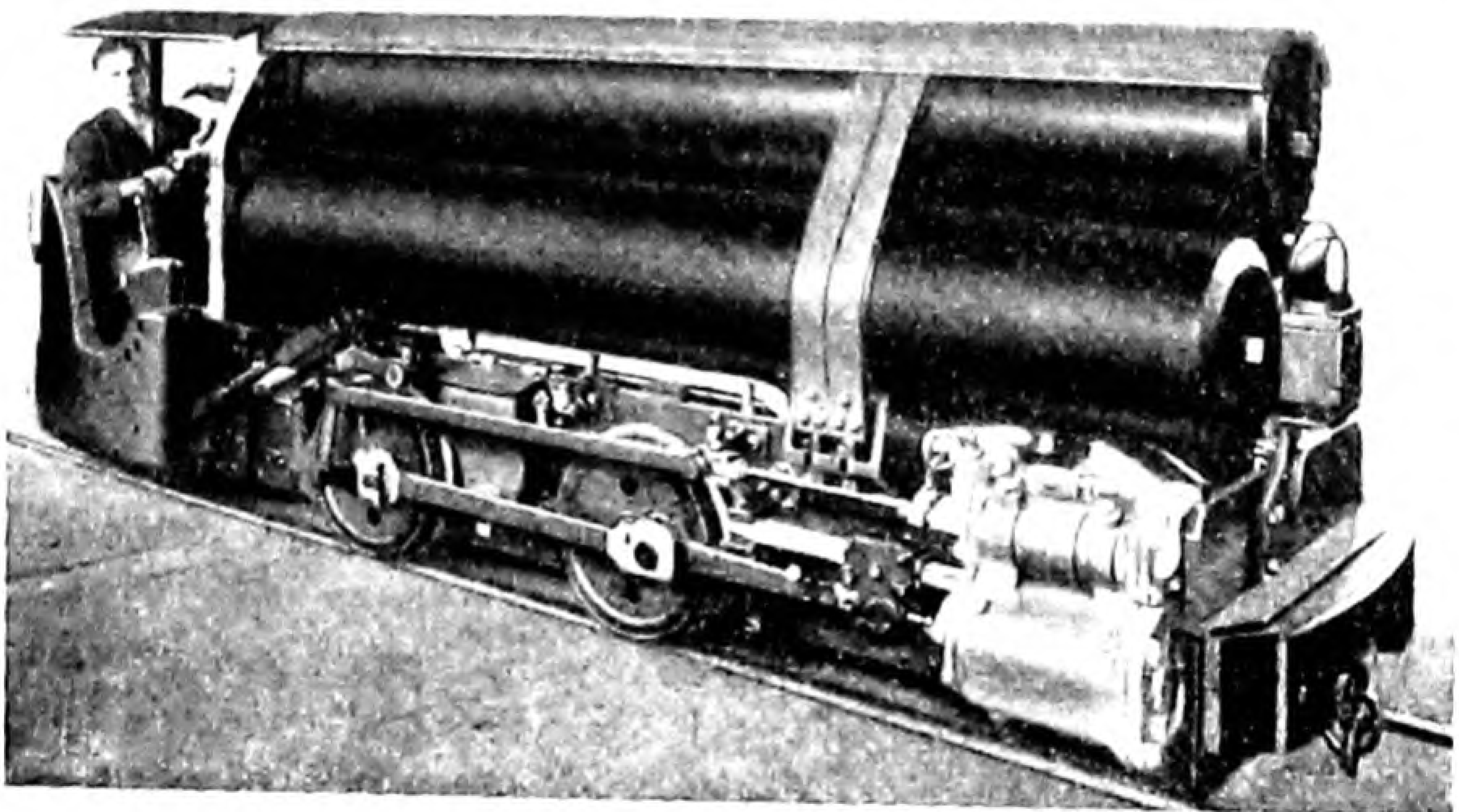


FIG. 321

gear. The total capacity of the reserve cylinders is from 150 to 200 cubic feet at a pressure of about 2,500 to 3,500 lb. per square inch. The air is compressed at the surface and brought underground through pipes. For medium capacities of from 400 to 1,000 cubic feet per minute, a vertical high-speed compressor, driven by an A.C. motor or high-speed steam engine, is usually preferred. Where 2,000 or 3,000 cubic feet per minute of compressed air is required, it is usual to have a five- or six-stage compressor running at about 100 r.p.m.

In order to determine the quantity of compressed air required, from 2·5 to 3 cubic feet per minute per locomotive horse-power should form the basis, according to the size of the locomotive used. The operational capacity of the compressor units should be suitably distributed, at least two units being in operation. A suitable arrange-

ment is to install three or four units, maintaining in either case one unit in reserve, and so having a reserve factor of 1·5 or 1·33. Thus, the compressor capacity required will be (1·33 to 1·5) (2·5 to 3) = 3·3 to 4·5 cubic feet per minute per horse-power of each locomotive. With this arrangement it is possible to increase slightly the number of locomotives in operation without increasing the compressor capacity. The compressed-air supply pipe in the shaft and the main roads is 2 inches in diameter and in subsidiary roads the range may be $1\frac{1}{8}$ inch in diameter. Where a 2-inch pipe is used in subsidiary roads, special receivers need not be installed. Changing stations have to be provided at appropriate positions along the supply pipe and, according to the length of the haul, the locomotive cylinders are recharged as often as required, but a reserve should always be retained.

Haulage costs with compressed-air locomotives. As in the case of other types of locomotive, a comparison must be made between the initial installation costs and the operational costs.

The initial costs can be expressed as follows:

$$C = \{[(1\cdot3 \text{ to } 1\cdot2)\ an\ \text{h.p.}_{\text{Loc}}] + [4bn\ \text{h.p.}_{\text{Loc}}] + [cl]\}\ \pounds,$$

in which (1·3 to 1·2) is a reserve factor depending on the number of locomotives, *a*, *b* and *c* are as indicated below, *l* is the length of the grid and *n* the number of locomotives.

The initial cost per locomotive horse-power *a* can be assumed to be £55 per horse-power for large locomotives and £30 for smaller locomotives.

The initial cost *b* of the high-pressure compressors per cubic feet per minute can be assumed to be about £9 for small units, £10 for medium-capacity compressors and £14 for larger units. The higher cost per cubic foot per minute for the larger unit is contrary to the general rule, but can be explained in this case by the fact that the smaller units are assumed to be driven by high-speed electromotors, the larger units by steam-engines. The assumed compressor capacity per horse-power of each locomotive has been taken as 4 cubic feet per minute.

The initial cost, *c*, of the compressed-air grid is about £2,750 per mile, but if additional receiver volume is installed the cost may be increased up to 50 per cent.

The total initial cost for ten 40-h.p. operational locomotives can be assumed to be from £90,000 to £100,000, or from £225 to

£250 per locomotive horse-power. The calculation of the depreciation cost can be based on a life of twenty years.

The operational wages cost will include the wages for two drivers per locomotive and additional wages for the attendance and maintenance of the locomotives and compressors. Compressor attendance can be assumed to be three shifts per day, while maintenance of locomotives and compressors can be taken at from 0·3 to 0·6 shifts per day.

The cost of spare parts for the locomotives varies within large limits according to the age and operational condition, and may be between £45 and £450 per locomotive per annum. An average spares cost can be taken as £140 per annum. To cover compressor spares and pipe-line replacements, about £25 per locomotive per annum should be added to this cost.

The consumption of lubricating oil is about 150 gallons per locomotive per annum, while the consumption of compressed air measured as free air at the compressor varies between 90 and 130 cubic feet per gross ton-mile.

The total operational cost of compressed-air locomotive haulage is higher than that of any other kind of locomotive haulage. The higher cost depends, in the first instance, on the lower efficiency of compressed-air operation and thus on the higher cost of power.

Section 5. Comparison of the Various Types of Locomotive Haulage

When deciding upon which type of locomotive haulage to install under certain conditions, every consideration must be taken with regard to ease in operation, safety and cost. The flexibility of operation must include readiness for service, limitations of travel and running conditions. Readiness for service in the case of diesel and compressed-air locomotives depends upon the supply of fuel oil or compressed air from storage tanks or from pipe-lines, while in the case of battery locomotives it depends upon the charging conditions for the batteries. In the case of trolley-wire locomotives, the locomotive is ready for service as soon as the rectifier unit is switched into operation. The limitation of haul or travel in the case of diesel, compressed-air and battery locomotives is dependent upon the supply of power carried by these locomotives, while with trolley-wire locomotives it depends upon the limit of installation of the overhead line. In the case of battery locomotives, the power reserve is normally sufficient for a single shift, and in the case of diesel

locomotives for two shifts. With large compressed-air locomotives the limit of haul, inbye and outbye, is from 1·5 to 2 miles, and repeated refilling during the shift is necessary.

It should be noted that electric locomotives are distinguished by their high overload capacity and by complete uniformity of torque at the driving wheels. These factors are not only important for ease in starting, but also for overcoming steeper gradients. With compressed-air locomotives, overload capacity is also possible by altering the cylinder cut-off. In comparison, diesel locomotives have a low overload capacity, but the tractive effort can be increased within the limits of the gear box, and so the engine gear box has to be designed in accordance with the greatest tractive effort required. The operational reliability of the trolley, battery and compressed-air locomotives is greater than that of diesel locomotives, the latter requiring more and special attention. From the safety point of view in operation underground, the compressed-air locomotive must be accorded first place. The only possible hazard may be the bursting of the air-storage cylinders and pipes, which must be inspected regularly.

Diesel locomotives require special appliances for protection against the danger of ignition of methane and for the elimination of carbon monoxide from the exhaust gas. These protective appliances require careful and constant supervision to be fully effective. Some comment has been passed on the smell of the exhaust gas and the possibility of confusion with the smell associated with underground heatings. The exhaust gas increases the temperature, the CO content and humidity of the mine air.

Battery locomotives must be of flame-proof construction but require less supervision and maintenance than the diesel locomotive and can be controlled more easily.

The operation of trolley-wire locomotives has the danger of accidental touching of the overhead line, the danger of current leakage, of fire and ignition of fire-damp. Satisfactory arrangements to eliminate these dangers can be made, except in the case of fire-damp ignition. The danger of fire-damp ignition can be reduced by minimising the production of sparks. In the Ruhr, the use of trolley-wire locomotives is limited to roadways which are clear of gas or where the methane content is less than 0·3 per cent. All locomotives transfer heat to the surrounding air, but the diesel engine transfers more than the electric types.

Considering the relative initial cost of locomotives and taking the diesel locomotive as unity, the initial cost of trolley-wire locomotive haulage is about 1·6, battery locomotive haulage from 2·7 to 3·2 and compressed-air locomotive haulage about 3·2. In the case of the battery locomotive, the range given covers the cost with and without reserve batteries.

When the operational cost comparison includes interest and depreciation, the trolley-wire locomotive cost is lowest, as in Fig. 322. If this cost is taken as unity, the battery and diesel locomotives show an interest and depreciation cost of 1·27 and the compressed-

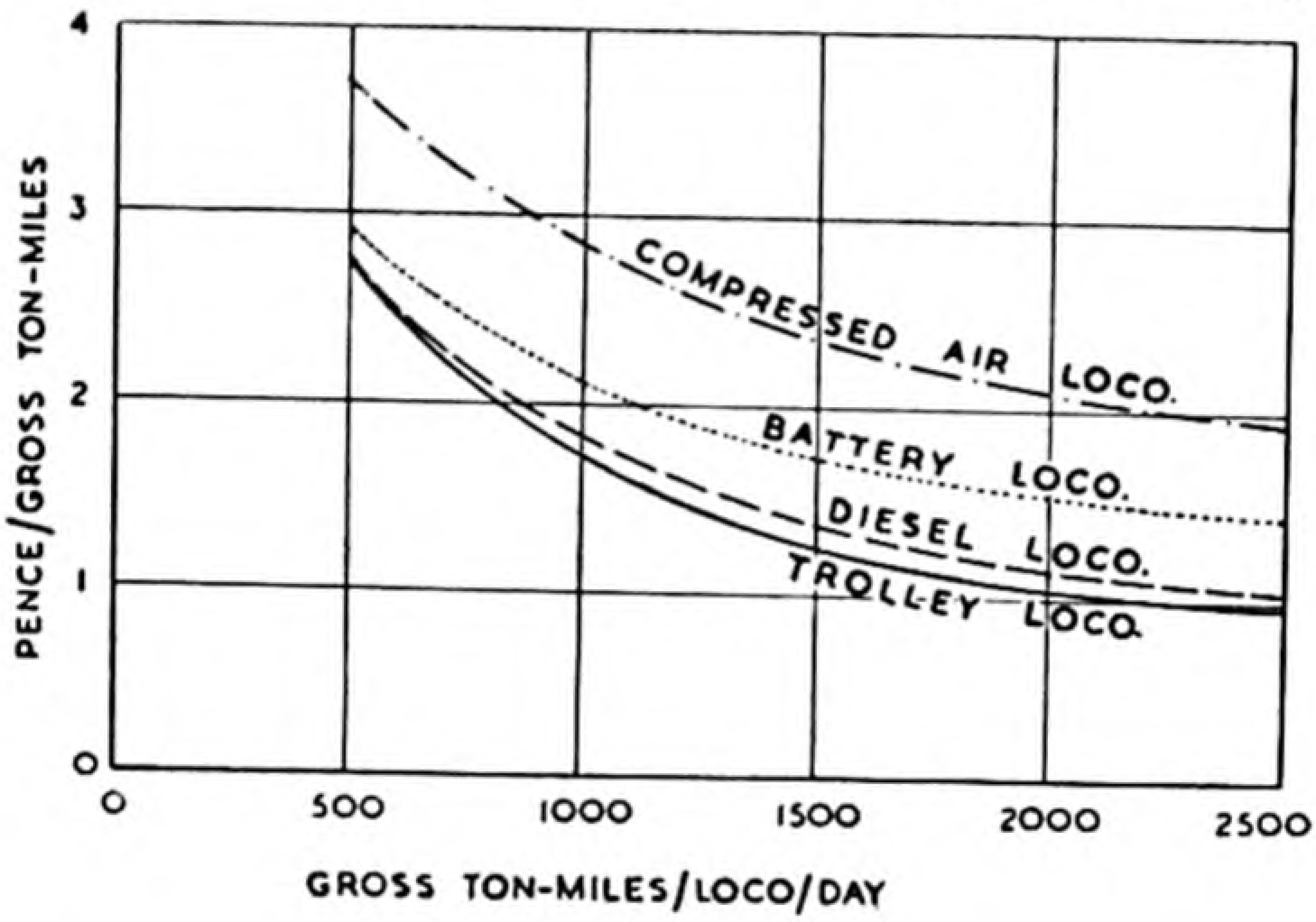


FIG. 322

air locomotive 1·35. The lower cost of operation of the trolley-wire locomotive is due to the lower repair and spares cost, and the relatively cheap and easily utilised power source. The high figure for the compressed-air locomotive is due to the high cost of energy, interest and depreciation. The operational cost of diesel and battery locomotives is approximately the same, as the higher capital cost of the battery locomotive is more than offset by its lower power, attendance and repair cost.

In each individual case, the cost of the locomotive haulage and the type of locomotive installed will depend on the operational conditions and mine organisation. These influences may be expressed numerically by considering the average daily load of the locomotive system. For this comparison it is recommended that the factor of gross ton-miles should be used and not net ton-miles.

Since the frictional resistance of the track system cannot easily be computed, the gross ton-mile based on purely statistical information can be regarded as a sufficiently accurate basis for comparison. Contrary to the net ton-mile, the gross ton-mile takes into account the transport of stores and waste as well as the loss in pay load due to the dead weight of locomotive and mine cars, which factors can be influenced by organisation of the system.

The number of gross ton-miles travelled per locomotive varies considerably from mine to mine, since it depends upon the distance to loading-points and panel stations from the shaft, on the influence of waste transport, lost time during gathering operations and other waiting periods. The average daily performance in the Ruhr for all main-road locomotives amounts to about 600 gross ton-miles. This value increases to more than 2,000 gross ton-miles in individual cases with favourable locomotive utilisation conditions, so that the ratio between gross ton-mile and net ton-mile varies between 2 : 1 and 3 : 1.

In general, trolley-wire locomotives have priority of choice in the Ruhr, where the danger from fire-damp remains within moderate limits, where the construction and maintenance of large and high roadways does not cause special difficulties, or where the height of roadway required for trolley-wire locomotives is already existent for ventilation reasons. The sphere of operation of the trolley-wire locomotive may be increased in conjunction with the use of battery locomotives. In such cases, trolley-wire locomotive haulage is limited to main roads free, or almost free, from methane, while the battery locomotives are used for other roads, thus serving as gathering locomotives for the trolley-wire locomotives. In mines where the haulage roads are widely dispersed, or where heavy strata pressure entails high road maintenance, or where danger from fire-damp explosions exists, diesel or battery locomotives are to be preferred. In addition, battery or diesel locomotives must be recommended for the transition period during development of the mine or levels, as they are independent of an overhead line, which is usually installed only after the drivage of a large section of the level drifts and coal working has commenced.

New installations of compressed-air locomotives will probably be considered only where there is any great risk of fire-damp explosions, because of their high operational cost.

Section 6. Track Construction and Layout for Locomotive Haulage

(*a*) *General introduction.* The proper construction of the track or permanent way is of primary importance for the efficient operation of locomotive haulage. The heavy weight of locomotives and cars in operation makes it desirable to give as much consideration to this specialised subject as is given in the case of railway engineering. A major part of the additional cost of main-road haulage will depend on the standard of installation and maintenance of the track. Poor sidings require heavier locomotives, reduce the life of the locomotives and rolling stock, increase the maintenance and power costs and reduce the haulage efficiency.

The track installation can be considered in two parts, the superstructure comprising rails, sleepers and switches, and the substructure consisting of the artificially made bedding for the superstructure.

(*b*) *The rails.* The rails used in normal rope-haulage practice are inadequate for locomotive haulage. The rails are of rolled steel of the same flat-bottomed section as those for surface railways, consisting of a head, web and foot. The rate of wear and tear depends on the hardness of the head, the bending strength depends on the height of the web and less on its thickness, while the thickness and width of the foot affect displacement and resistance to tipping.

The choice of the size of the rail section is of fundamental importance, the rails being classified according to their weight in pounds per yard and/or to their height. The existing regulations in force in British mines specify a minimum weight of rail which is related to the axle load of the locomotive. Thus the minimum weight required is 10 lb. plus 5 lb. per ton of axle load, with a minimum of 40 lb. per yard. The National Coal Board Traction Sub-Committee recommend a minimum weight of 50 or 60 lb. per yard, suitable for axle loads up to 8 and 10 tons respectively. The weights of main-road locomotives vary between 7 and 15 tons according to their capacity, and the axle load between 3·5 and 5 tons. The axle loads of small mine cars are less than 0·5 ton, but may increase up to 2 tons with high-capacity mine cars. Other factors have to be taken into consideration with main-road haulage, including the additional stresses caused by floor movement, humidity of the mine air and the corrosive effect of mine waters. For these

reasons, heavier sections are normally chosen than would be required by the axle pressures of from 3·5 to 5 tons. In some cases, much lighter sections are suitable for gate roads and sub-level cross-measure drifts. The N.C.B. Traction Sub-Committee recommend that a 28-lb. section may be used only with a maximum locomotive weight of 5 tons in a road used only for supplies or stores. The graph, Fig.

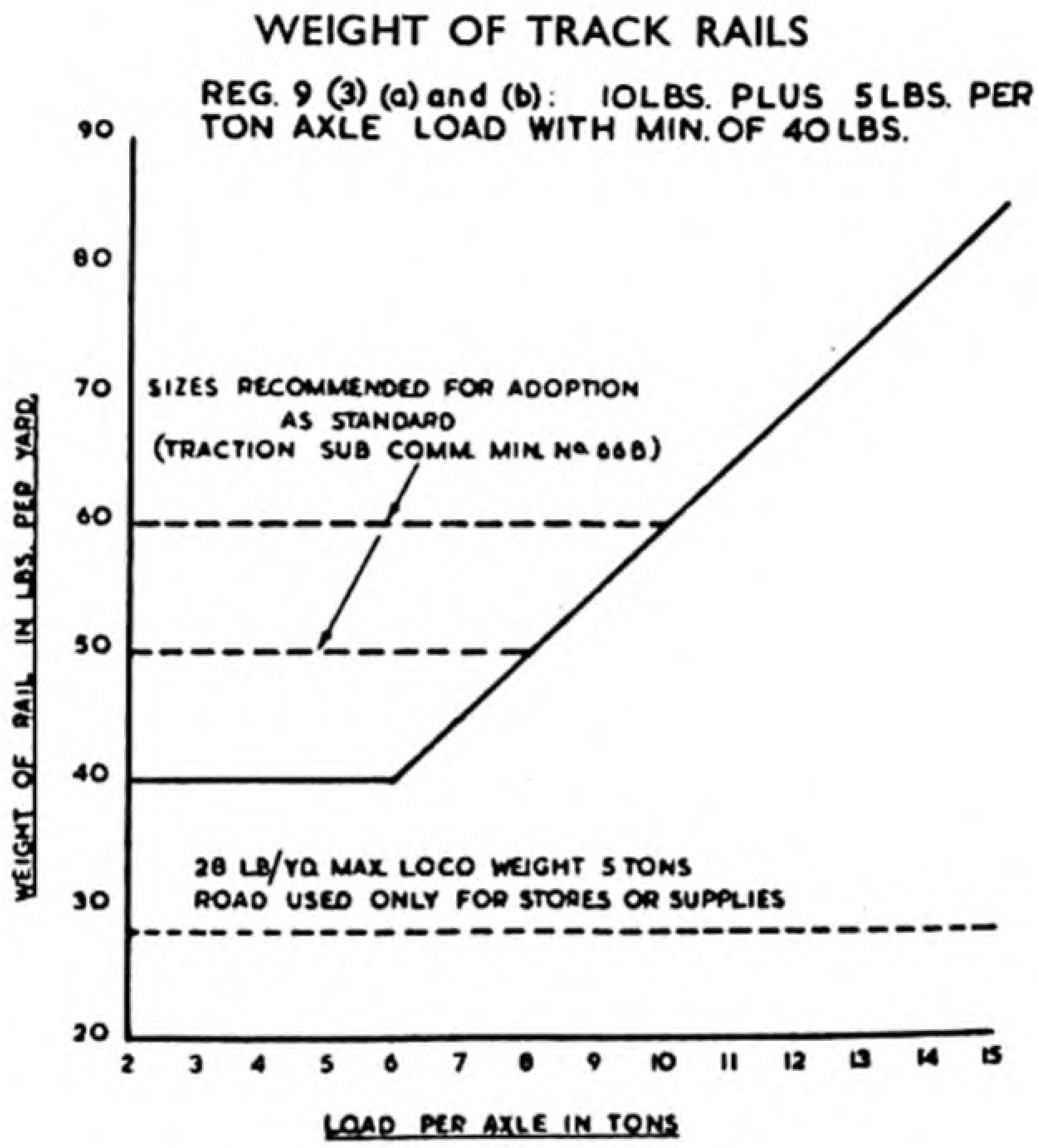


FIG. 323

323, shows the weights of track rails and axle load relationship as recommended by the N.C.B. Traction Sub-Committee.

(c) *Switches*. The control of the movement of trains from one track to another is carried out by switches, which may be either left-hand or right-hand. Where the train passes over one track into two parallel tracks, symmetrical switches are used. The connection of two parallel tracks is done with transit switches, consisting of two right-hand or two left-hand switches.

The switch tongues, which are movable, guide the wheel flanges in the appropriate direction. The tongues may be moved by bars or by the action of levers from a frame on the side of the switch, or by small lateral pneumatic cylinders operated automatically.

Special switch constructions are represented by fixed switches

and spring switches, both of which are frequently used at the pit bottoms of main or staple shafts. A fixed switch is used where the trains move constantly in one direction. In this case the switch tongue is fixed, whereas in the spring switch two directions of movement can be accommodated.

In locomotive haulage, automatic switching is extremely important. This feature can be incorporated in the system by installing a pneumatic ram which controls the switch. The pneumatic ram

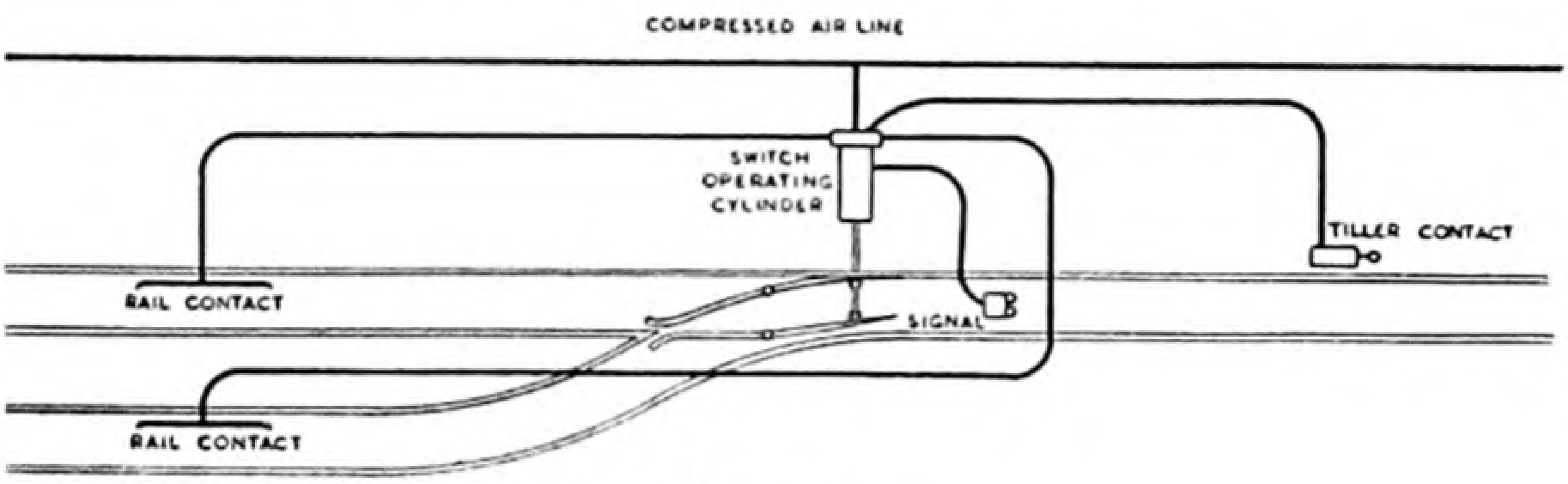


Fig. 324

may be operated by the driver or by the locomotive tripping a catch. A semi-automatic switch operated by the driver is illustrated in Fig. 324.

(*d*) *Sleepers.* The sleepers may be of timber or steel, the former being most commonly used. Timber such as oak, which is hard, tough, sufficiently resistant to a humid atmosphere and will permanently retain spikes and bolts, is best suited for main-road track. Beech or coniferous timber may be used for subsidiary and gate-road haulage track. The dimensions of the sleepers will depend upon the load the track is to carry. For main roads, the sleepers are generally $4\frac{1}{2}$-inch by 12-inch section, and for gate roads 4-inch by 5-inch or 3-inch by 4-inch are satisfactory.

Steel sleepers may be used in dry roads where the floor movement is small or non-existent. These sleepers rust easily and are attacked by acid mine waters. Galvanising offers some protection against rust but considerably increases the cost. Bedding steel sleepers into the track penning is more difficult, since they are usually of hollow section.

The interval between sleepers depends upon the load, the nature of the ground and the life of the roadway in which they are being installed. In main roads the distance varies from 1 foot to 2 feet 8

inches apart, but in the case of crossings and switches this distance may be considerably reduced. The distance should also be reduced at rail joints, a particular precaution with locomotive haulage, but the distance apart should be sufficient to allow under-filling and tightening up of the penning between the sleepers. The maximum distance apart can only be calculated taking into account the axle load, the rail-section and the sleeper width.

In the case of double track, sleepers should be installed which will span the width of both tracks. Where heaving of the floor is anticipated, it is better, however, to install each track on separate sleepers, so that track-levelling operations can be carried out separately.

(*e*) *Rail fastening and track bedding*. Reliable methods of rail fastening become more important with locomotive haulage, the greater the capacity of the mine cars used. The tying must be carried out in such a manner as to protect the sleepers against pressure from the rails, to avoid damage to the spikes or screws in the event of a derailment and so that replacements can be made easily. The ties must offer sufficient resistance to horizontal and vertical forces. Horizontal forces are set up by the pressure of the conical rolling circles of the wheels, especially when driving round curves, while vertical forces are introduced by the bending of the rails between the sleepers. In these circumstances the rail ties are subjected to alternating compressive and tensional stresses.

Where rails are being fastened on wooden sleepers, spikes are usually preferred, since it is believed that the driving in or extraction of the spikes is easier and the spike heads will suffer less damage than screw-heads through derailments. The adhesive strength of the screw is, however, greater than the spike, which is reduced through time, while in the case of the screw, it can even increase. The track-laying cost with screws is not more than with spikes, if the same standard of work is demanded in both cases. Preboring of the sleepers should be carried out, preferably on the surface. The best results have been obtained with cylindrical screws, and conical screws should not be used. The screw-head should be low and wide, and either square, rectangular or hexagonal heads may be used.

In order to protect the head of the screw or the spike and to give a larger hold on the rail foot, clamping plates may be used. A disadvantage of these plates is that they wear easily, rust and affect the grip of the screw on the rail foot. It is preferable to have a screw

with a head large enough to provide an adequate hold and with a strong shank.

Washers are used to assist in distributing the load of the rail foot over a wider area of the sleeper and to assist in resisting lateral loads. Ribs and hooks on the lower side of the washer are used to provide a stronger adhesion between the washer and the sleeper, though

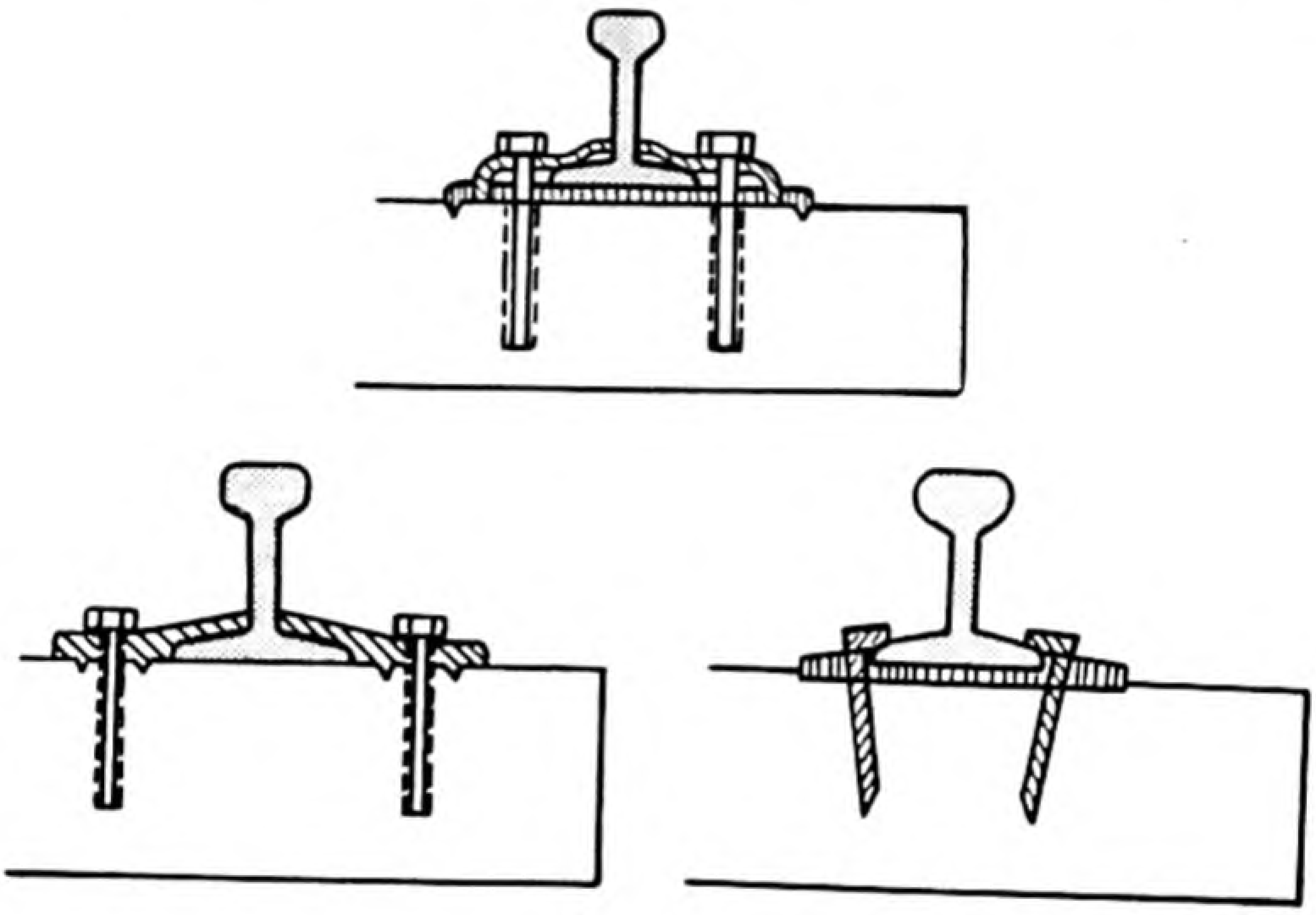


Fig. 325

these may cause a more rapid deterioration of the timber. The best method is to use a strong rail with a wide rail foot and a wide sleeper of hard timber.

In straight roads with undisturbed floors and medium axle loads, the previous means of fastening will be sufficient. At curves and at rail joints, as well as in roadways, where strata pressure is creating an effect, shoes should be used to provide a reinforcement. It is important to have more than one hole in the shoe. If only one hole has been provided on each side of the rail and if the screw or spike becomes loose, the joint will be worthless. The loosening of the joint will be easily transferred to other sections unless another hole is provided and an additional screw driven into the sleeper. Several methods of rail fastening are illustrated in Fig. 325.

Opinions differ regarding the provision of protection of rail ties or joints from possible derailments. The best means is undoubtedly to avoid derailment by strong track construction, and this is more

important the greater the capacity of mine cars in use. In roads where floor movement is occurring, this objective cannot be realised and apart from mine-car re-railing devices, as shown in Fig. 326, protection should be given to the heads of spikes and screws. This can be done by countersinking the screw-heads or by providing lip-ledges to the washers.

The type of rail joints is extremely important, since they are the weakest part of the track and cause most track damage. It is essential

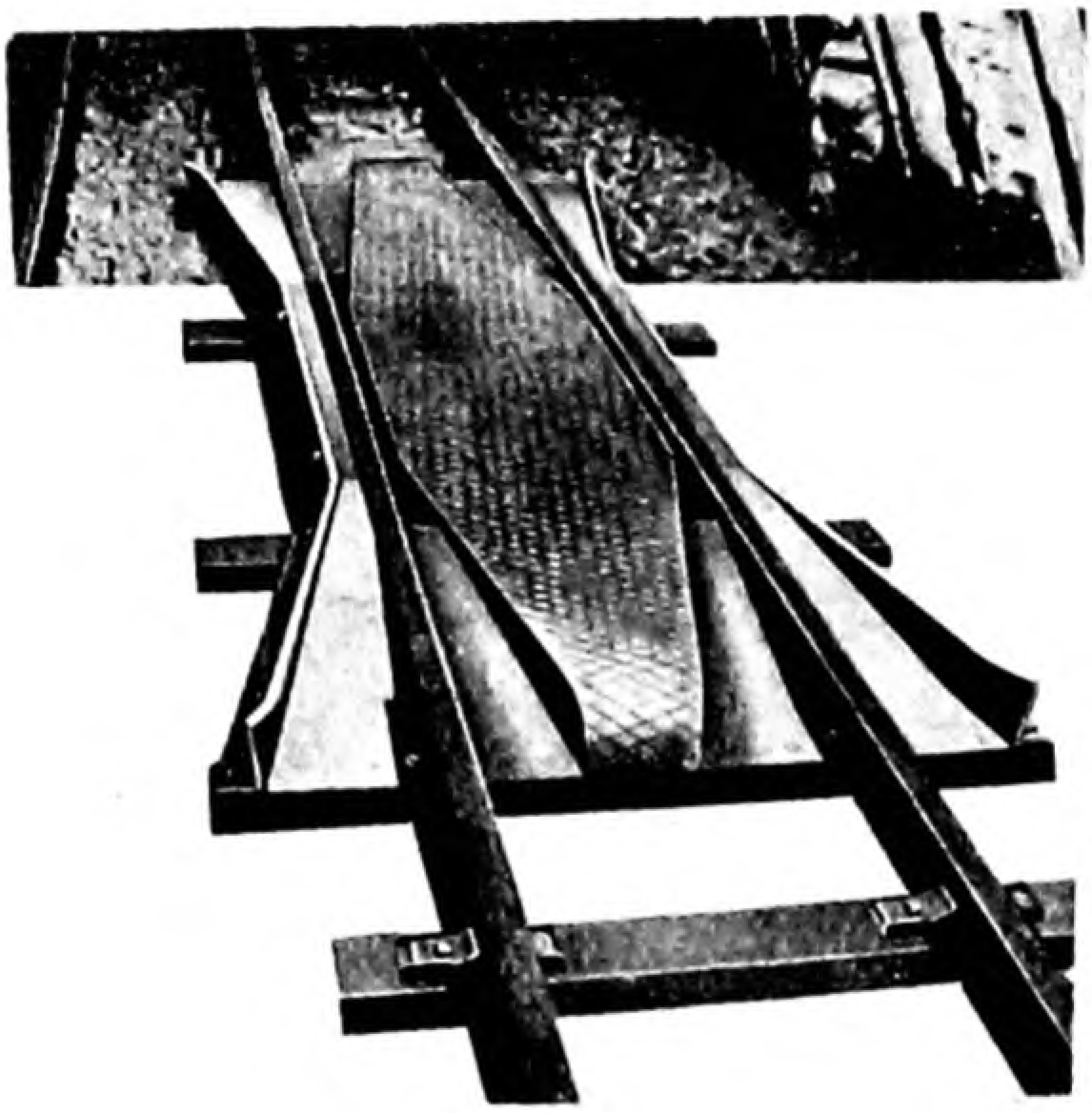


FIG. 326

to reduce the number of joints to the minimum by utilising long rails or by welding, though the latter is not always possible. The best joint is laid on a wide sleeper or double sleeper with a common, strong shoe, as shown in Fig. 327. In roads with floor heave, the joint should be made in such a manner as to allow longitudinal extension, by providing enlarged holes in the fish-plate fastening.

As well as the fixed joints, the free joints between two sleepers should also be noted. These free joints, as illustrated in Fig. 328, are being widely used, due to the ease in fitting the fish-plate and the reduction in shock to the haulage.

The fish-plates used are either the flat or angle type as shown in Fig. 329. Angled fish-plates are more rigid and are being used where the heavier loads are being carried.

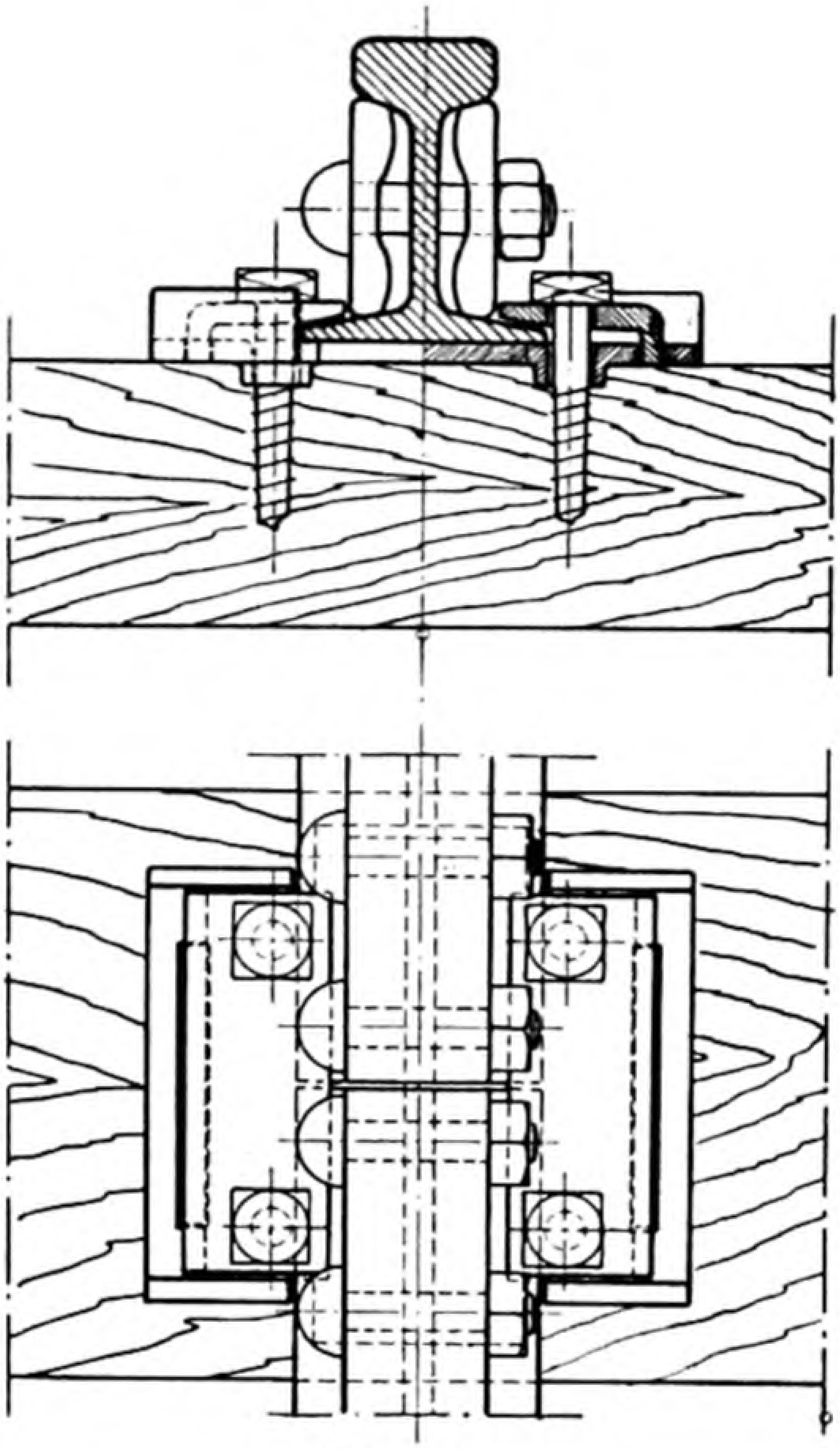


FIG. 327

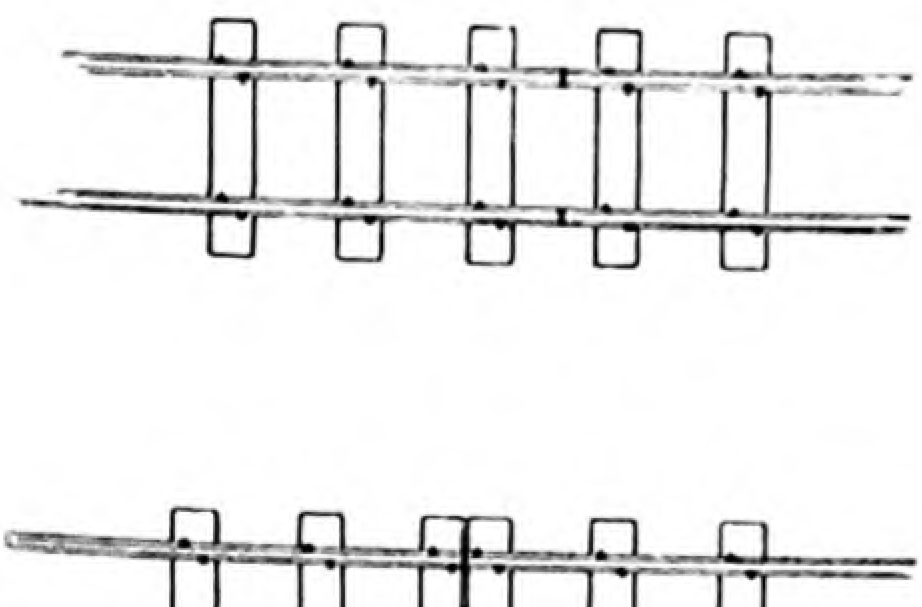


FIG. 328

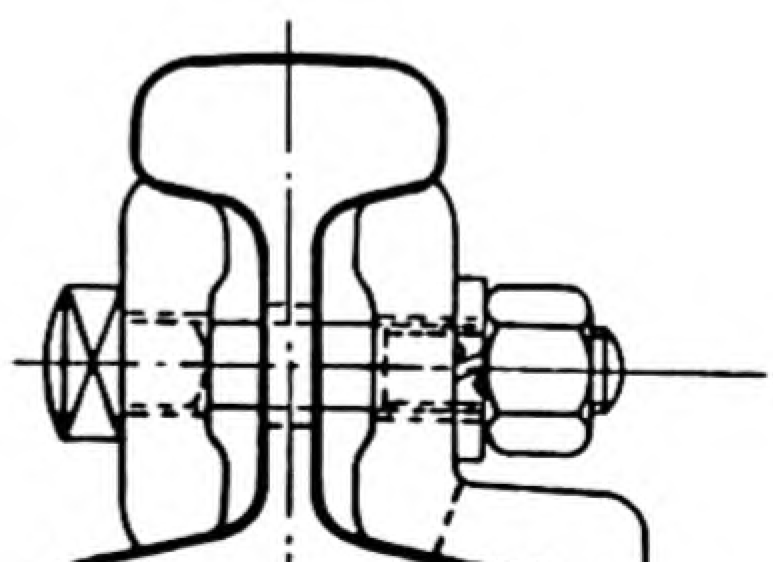


FIG. 329

The rail joints in a straight length of track should be opposite each other, whereas on narrow curves the joints should be displaced by about half the length of a rail. The position of the joints between the straight track and the curve should then be equalised by adjusting lengths.

Two further points must be considered with respect to curves, i.e. the centrifugal effect of the mine cars as well as the increase in the resistance to motion of the cars due to their deflection from their original direction of motion. The centrifugal effect is met by the super-elevation given to the outer rail of the curve. In locomotive haulage the degree of super-elevation can be computed from the formula:

$$h = \frac{1{\cdot}67v^2}{R},$$

where h is the elevation in inches, v is the maximum speed in miles per hour and R the radius of the curve in feet. It must be considered that the speed of underground trains on curves must be considerably less than when on straight roads. Taking as an example, $v =$ 7 m.p.h. and $R = 50$ feet, the elevation $h = 1\frac{5}{8}$ inches.

The deflection-resistance factor can be reduced by increasing the track gauge in curves by about $\frac{3}{4}$ inch, compared to $\frac{3}{8}$ inch on straight roads. An approximate value is that the radius of the curve should be at least ten times the wheel base.

The position of the curves and switches with regard to the direction of the motion of the load should be mentioned. The track layout should be carried out in such a manner that the loaded trains should have to pass the minimum of deflections and curves. The correct and wrong ways of the development of track layout are shown in Fig. 330.

The sleepers can only be laid directly on the floor in gate roads. In the case of locomotive haulage in main roads, however, it is necessary to take similar measures as those taken on surface railways to provide sleeper bedding. The bedding or ballast has to take up the pressure of the sleepers and transfer it to the floor. The bedding can also assist in compensating for changes in floor level as well as protect the sleepers from humid conditions. Ballast material can be taken from rippings in headings for straight road track. At wet sections or where load shocks are likely to arise at joints or in curves,

ballast from some other source is advisable. The cross-section in Fig. 331 shows a road with suitable track ballast and lateral drainage. Lateral drainage should be provided with an inclination of 1 in 50 towards the drain at the roadside.

(*f*) *Track maintenance.* The main work associated with mainten-

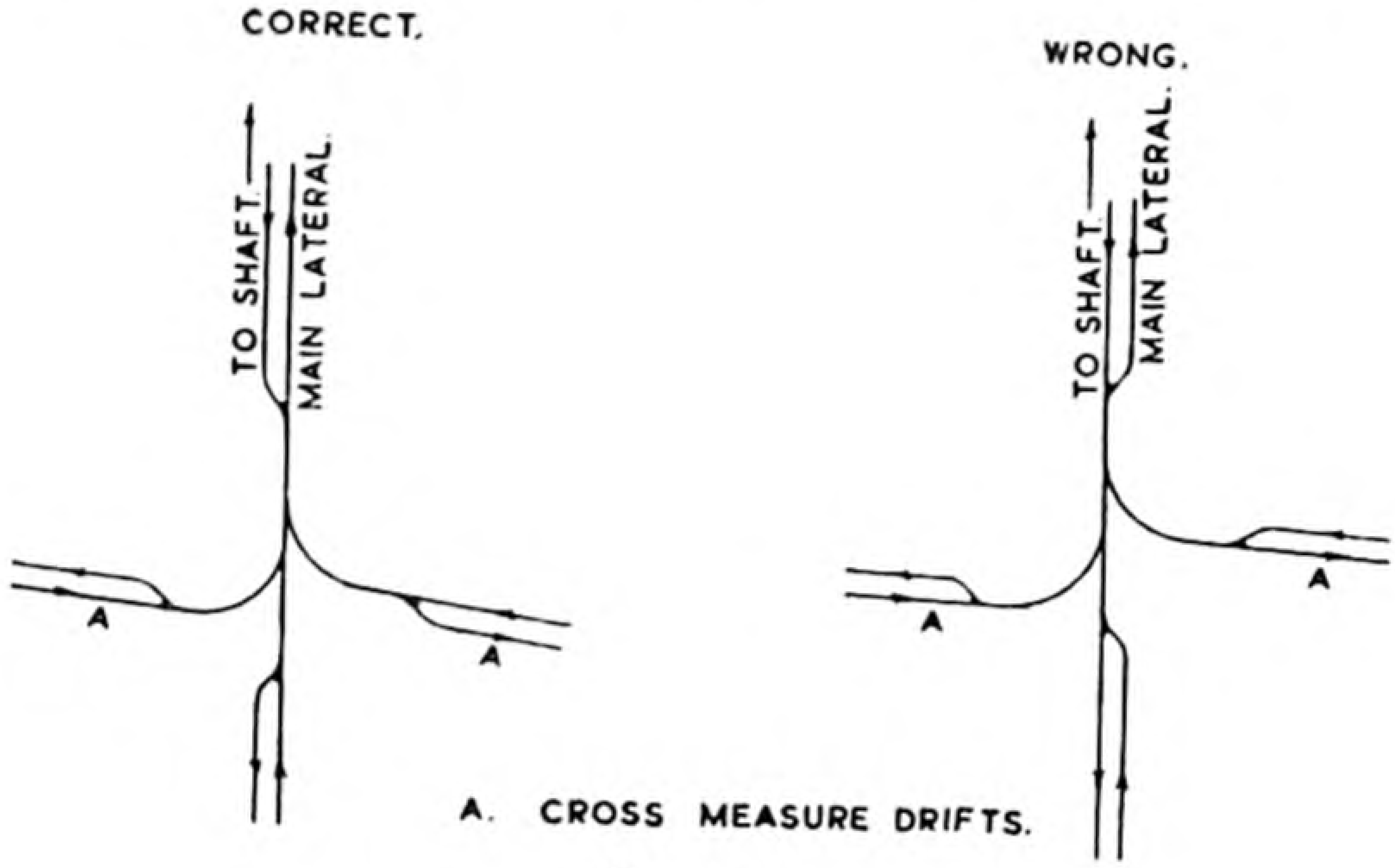


FIG. 330

ance is lifting joint sleepers and the redriving of loosened screws, etc. It is important to have the maintenance work on the track carried out as far as possible by the same workers. Small repairs should be carried out immediately so that the effect cannot be

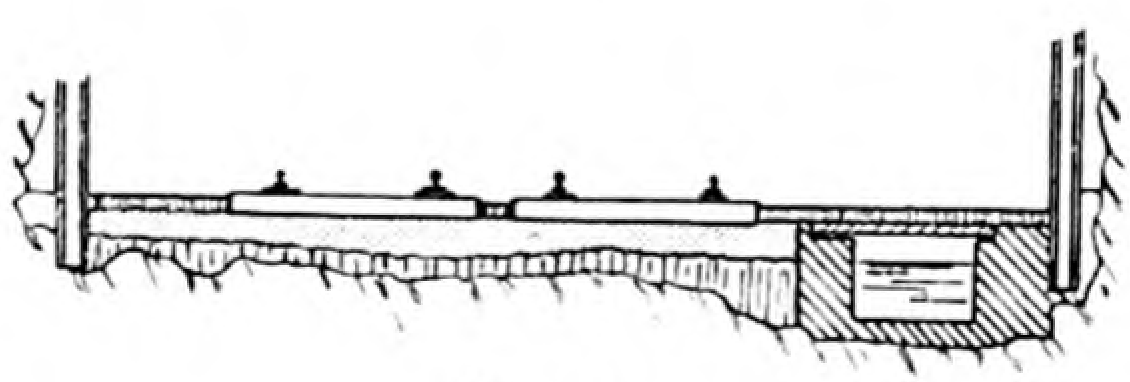


FIG. 331

intensified and transmitted along the track. The need for repairs should be discovered at an early date. The only safe means is to find out these places by observation from the locomotive, especially the smoothness of action and conditions at joints. The best personnel for this task are the engine drivers, who should be instructed to indicate and immediately report damaged track sections.

Section 7. Mine Cars

(*a*) *General introduction.* A general distinction can be made according to the capacity of the mine cars employed in German mines, where small cars have a capacity of up to 1,000 litres (35 cubic feet), medium cars, from 1,000 to 2,500 litres (88·5 cubic feet) and large mine cars, over 2,500 litres. In addition to these, there are in use a number of mine cars of special design. The National Coal Board have approved and issued a standard specification for solid-bottom mine cars (P.D. No. MM. (49) 1.—SMRC. 1949) which refers to mine cars having nominal coal pay-loads of from $1\frac{1}{2}$ to $3\frac{1}{2}$ tons and volumetric capacities of from 61 to 142 cubic feet.

While mine-car haulage had formerly dominated the field of underground transport in coal-mining, the development of continuous haulage methods has reduced the scope of the application of this form of transport. With the exception of bord-and-pillar working, the mine car has almost completely disappeared from auxiliary haulage near the coal face, while their use in gate-road transport has been considerably reduced. On the other hand, however, the importance of main-road haulage and the necessity for a regular cycle of operation has considerably increased the importance of mine-car transport, particularly in large developing collieries with extensive roadway networks.

The rolling stock at a mine using mine cars for main-road haulage represents a very large capital expenditure. At an average price of, say, £150 for a $2\frac{1}{2}$-ton-capacity mine car, the rolling stock required for a mine with a daily output of 4,000 tons would be approximately £45,000. Thus, the choice of the type of mine car and the estimation of the number required is of extreme importance.

(*b*) *General requirements for mine cars.* There are many general requirements which have to be fulfilled for a mine car, and these are often contradictory in terms. They should be cheap, have a small weight together with a large capacity, i.e. a favourable ratio between dead load and useful load, and a long service life. In operation, the cars should have a low resistance to motion, keep to the track safely and travel easily through curves. The cars must be easily manœuvrable and retracked after derailment, while a factor of great importance is that they should be easily emptied and cleaned. The mine cars chosen should be adapted to the particular mine conditions,

such as roadway and shaft dimensions, and in the case of steep-seam working, to staple-shaft cross-section and gate-road measurements. In these respects it is possible also to take mine-car dimensions into account, and a compromise has to be made between many of these requirements. Thus, for instance, low cars will be steady and easy to load and haul. In order to obtain sufficient capacity, they should be long enough, and this may require larger shaft cross-sections. Cars having a wide wheel-base are particularly steady, but will travel less easily through curves. On the other hand, cars with a narrow wheel-base which travel easily through curves and can be retracked with little effort after derailment, tend to be unsteady and easily subject to derailment. Thus, even if all the requirements for the ideal car to suit the particular mining conditions cannot be fulfilled, an attempt should be made to find a solution which satisfies the installation conditions.

(*c*) *Minecar design*. The *small mine car* or tub consists of the mine-car body with buffers and the set of wheels and couplings. The body and wheels may be mounted on a special underframe. In Germany, the car capacity is expressed in litres and to convert into corresponding units of weight the following bases are used; 0·80–0·85 kg./l. for clean coal, 1·0 kg./l. for run of mine coal and 1·2–1·9 kg./l. for waste. The following table shows the capacity, dimensions and tare or empty weight of some German mine cars.

Type of Car	*Pay Load*	*Length*	*Width*	*Height*	*Tare*
	litres	*mm.*	*mm.*	*mm.*	*Kg.*
DIN Berg. 550 . .	750	1,700	800	1,000	535
,, ,, 551 . .	875	1,800	801	1,075	616
,, ,, 552 . .	1,000	1,900	802	1,150	720

l. ÷ 28·32 = cu. ft.; mm. ÷ 25·4 = inches; Kg. × 2·2 = lb.

The car body is usually of galvanised steel sheet, from 3 to 5 mm. thick. Formerly the body was of riveted construction, but in recent years the practice has been for welded construction. By this method, the weight as well as manufacturing cost and maintenance is reduced by about 10 per cent. The body edge is usually strengthened with a welded inside edge reinforcement of bar section. As a protection against hand injuries, pockets are generally arranged in the upper front and rear part of the body, while the token or number plate for the car is fastened with wire through two small holes.

The car buffers are simple steel head-sheets welded on to the front and end of the car. Separate buffers would be an advantage, but in the smaller German mine cars the additional cost is considered to make them unnecessary. Attempts have been made recently to adopt a light metal construction. Previous trials of this nature gave a 40 per cent. saving in weight, but the cost of the car was increased from six to eight times. The light metal cars were less resistant to corrosion, and the extended development of this method is dependent upon the development of suitable light alloys. On the other hand, cars with cast-steel bodies did not find favour because of their increased weight.

With regard to safety in operation and easy manœuvrability, the design of the wheels, axles and bearings is extremely important. The developments in this direction have been well advanced. Four different methods of construction of wheel sets can be considered:

(1) The wheels are keyed to the axles, which turn in bearings. This design, however, is rarely used today, due mainly to the effect of the outside wheels being braked on curves by the inside wheels because the former have a longer distance to travel. Friction and wear are therefore increased unnecessarily.

(2) The wheels turn on fixed axles. The disadvantage of this design is the difficulty in lubricating the friction-faces between the axles and the wheels and the protection of these faces from dust.

(3) The design in which (1) and (2) are combined in such a manner that one wheel is running loosely on each axle while the other is fixed to the axle. This method is often adopted.

(4) In this final design, the methods used in (1) and (2) are combined so that all the wheels and axles run loosely in bearings.

The two main dimensions on the wheels set are the distance between the axles or wheel-base and the distance between the inside face of the rails or 'track gauge'. Within certain limits the wheel-base must be increased with the length of the mine car to counteract any tendency to rocking and the possibility of derailment. On the other hand, this length must not be too great, or binding will occur when travelling through curves. In Germany the wheel-base for small mine cars is normally 475 mm. (19 inches). The track gauge, which will also increase with the width of the mine car, should also

be kept to a reasonable width to prevent binding through curves. On the Ruhr, the track gauge varies from 500 to 600 mm. (20 to 24 inches). Due to the heavy service conditions, the axles are usually of steel and the supporting bearing surfaces hardened. The bodies are made of tough, malleable cast iron or annealed cast steel. The car wheels must be especially strong and because of their small diameter, from 350 to 400 mm. (14 to 16 inches), they make a relatively large number of revolutions and are subjected to a great deal of wear and tear. The wheels are usually made from annealed steel, which is much more resistant than cast iron. The essential parts of the wheel are the boss or hub, wheel-tread and flange. The boss should not be too narrow, in order to distribute the wheel-pressure over as large a bearing surface as possible. The wheel-tread must be slightly conical so that it presses against the interior face of the rail, thus preventing lateral rolling of the cars and the consequent wear on the flange. Track-keeping should thus be mainly effected by the wheel-tread itself and only taken over by the flange in case of possible derailment and when travelling over switches and through curves. The boss and wheel-tread are connected by either spokes or discs, thus distinguishing between spoked wheels and centre-disc wheels. The spokes are slightly curved to provide a certain degree of elasticity in the event of vertical thrust. With centre-disc wheels, the wheel disc should be provided with four or six holes for weight reduction. With this type of wheel it is possible to avoid casting stresses and to install brake blocks if required.

Originally, closed sleeve bearings were adopted with fitted grease boxes. The axle bearings were, in some cases, metalled or sleeved with synthetic resin material. The present practice is to provide rolling friction and roller-bearings, which gave very satisfactory results in underground tests and are now in general use in German mines. The simple roller-bearing in which the rollers in their housing are arranged parallel to the axles has given way to the taper roller-bearing wheel. This type of bearing is mainly used for the larger mine cars, while in the case of small cars the self-oiling sleeve bearings are generally used.

A special under-frame connecting the body to the wheel-set is only used with trough-shaped, small mine cars, while with box-shaped bodies the base is set directly on the axles.

The advantage of an under-frame is to give a certain flexibility

between the body and wheel-set, with the possibility of providing a bumper mechanism on the ends of the car under-frame which will not affect the car body. Buffering on the under-frame has, however, the disadvantage that it may lead to swinging of the body on the under-frame and loosening of the connection. The provision of an under-frame increases both the weight and cost of the car, and reduces its stability due to raising the centre of gravity of the car.

The car couplings must not sag too far and foul the sleepers or switches, or catch projections of the safety blocks in inclined roads and at the shaft. The couplings should not project beyond the ends of the cars. They should be flexible in curves, but should not allow

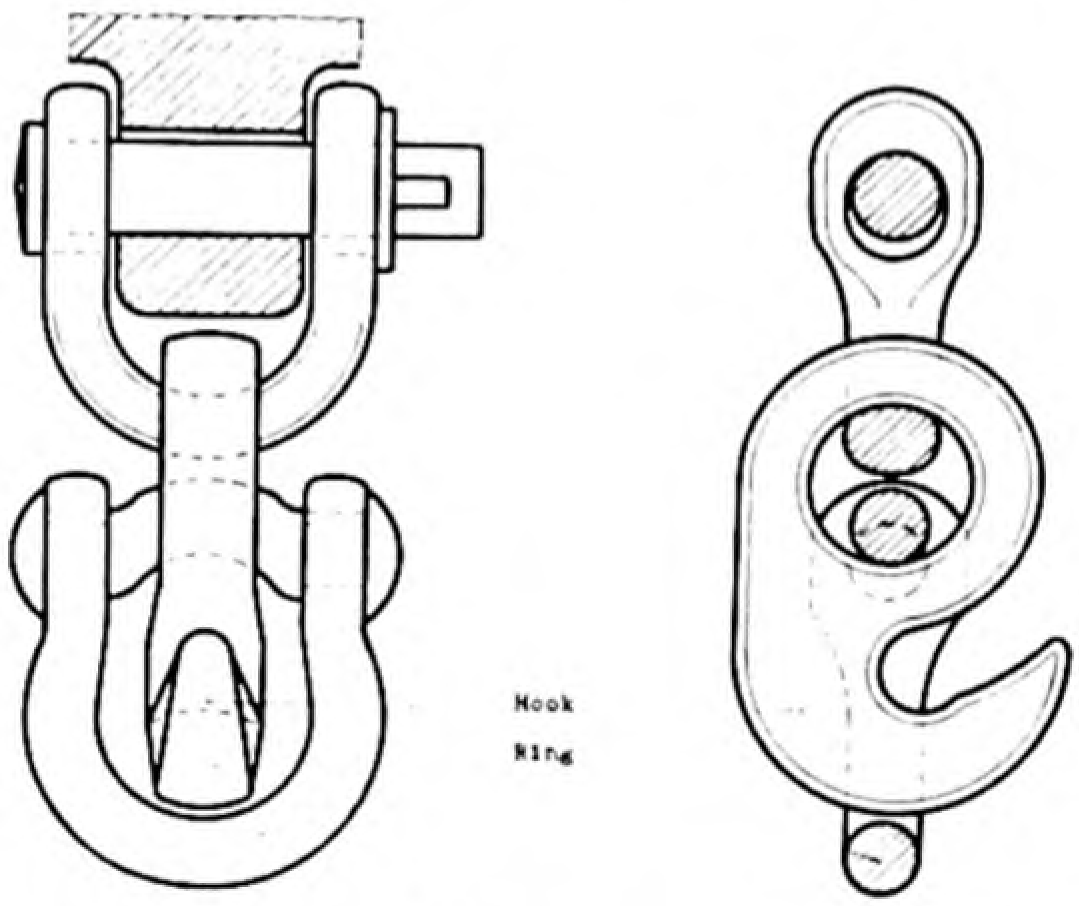


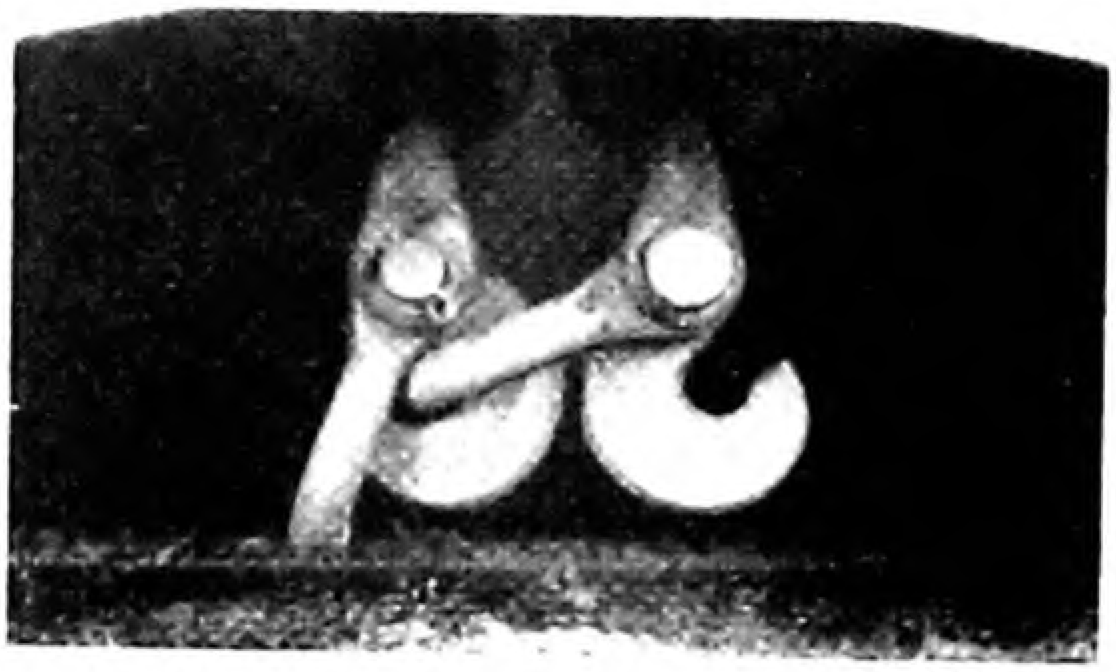
Fig. 332

too much freedom to cause jerking when starting but keep the cars close-coupled and together. The coupling should not retard positive catching of the car by a chain feeder, and they should be cheap, durable and easily replaced.

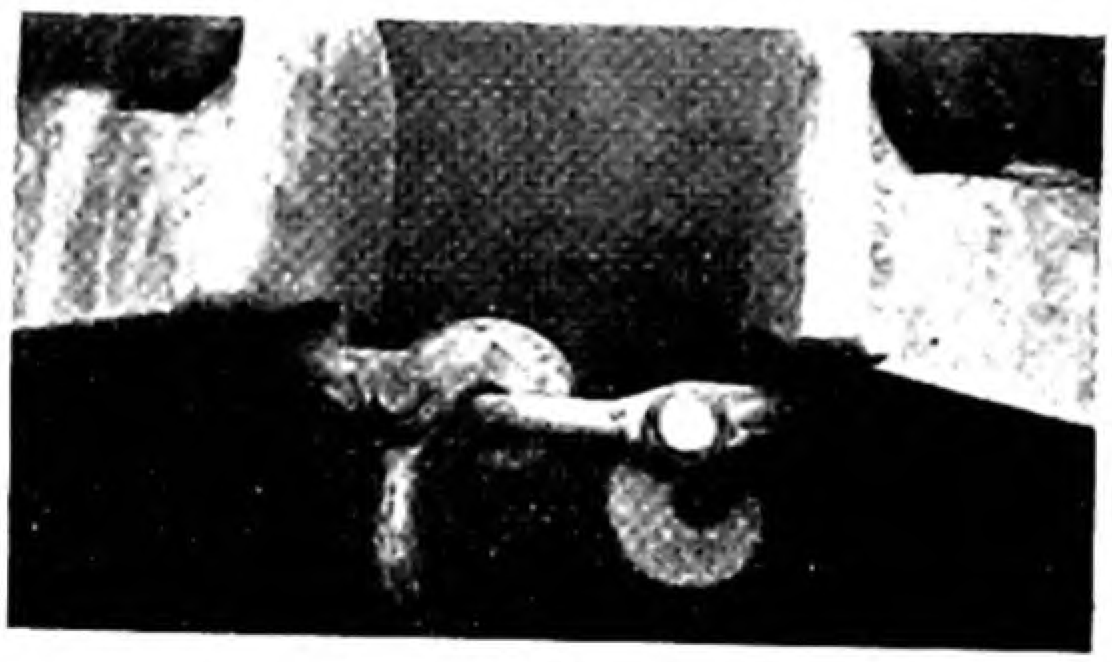
The coupling which is almost exclusively used for small mine cars is illustrated in Figs. 332 and 333. The action of the coupling is illustrated in Fig. 333, in which the close coupling can be seen. It is always advisable to put the hook of the following car in the ring of the preceding car so that uncoupling by fouling a sleeper is avoided.

The *medium and large mine cars* are essentially of the same general construction as the small mine car, but under-frames are invariably incorporated.

During the development of large- and medium-capacity mine cars in Germany, by enlarging the dimensions of the car, both long



(a)



(b)

(c)

FIG. 333

and wide cars have been produced. The following table gives the main details of long cars standardised in Germany and two wide-car types in use on the Continent.

Type of Car		*Capacity*	*Length*	*Width*	*Height*	*Tare*
		litres	*mm.*	*mm.*	*mm.*	*Kg.*
Long car M1600	.	1,600	2,500	840	1,250	850
" " M2000	.	2,000	2,760	900	1,300	950
" " M2400	.	2,400	2,940	970	1,350	1,050
Long car G3000	.	3,000	3,320	1,050	1,400	1,520
" " G4000	.	4,000	3,720	1,140	1,500	1,600
" " G5000	.	5,000	4,000	1,240	1,600	1,840
Wide car Type 1	.	3,000	3,010	1,500	1,150	1,935
" " Type 2	.	3,900	3,452	1,600	1,262	1,500

While the wide mine car predominates in the U.S.A. because of the different mining conditions, the long car has been successfully introduced into the Continental horizon-mining system. The long mine car allows full use to be made of the shaft cross-section and the provision of two winding sections in the same shaft, as shown in Fig. 334. Even if this advantage has not to be considered where skip winding is in use, it should be noted that this type of car is more adaptable to the use of narrower roadways in which double track may be installed, with the resulting saving in roadway maintenance and support. These advantages are opposed by the main disadvantage of the long car, which has a high resistance when travelling through curves and therefore must be driven with a sufficient radius. It should be remembered in this instance that individual cars will travel more easily through narrow curves than when coupled as a train, a factor which is important when considering the installation of 'roundabouts'.

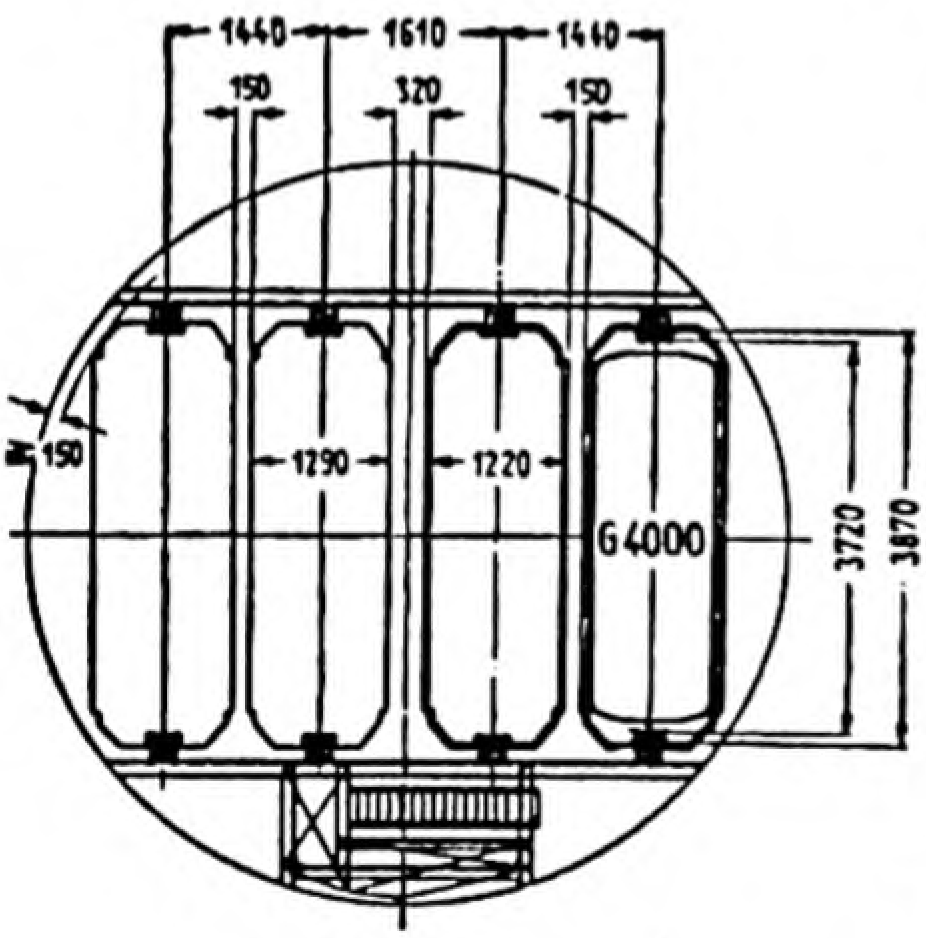


Fig. 334

Experience has shown that the most favourable width of car is 1,100 mm. (3 feet 8 inches) and that the height should not exceed 1,600 mm. (5 feet 4 inches), in accordance with the normal dimensions of gate roads, ease in loading, where necessary by hand, and to minimise the obstruction of the roadways for ventilation. A large-

capacity mine car of 3,000 litres capacity (106 cubic feet—2½ tons) is shown in Fig. 335.

The car body is exclusively of welded construction from galvanised steel sheet, 5 mm. thick (0·2 inch). The medium-car body is usually trough-shaped, while the larger mine cars are quadrangular. The ends of the larger cars may be chamfered where large and small

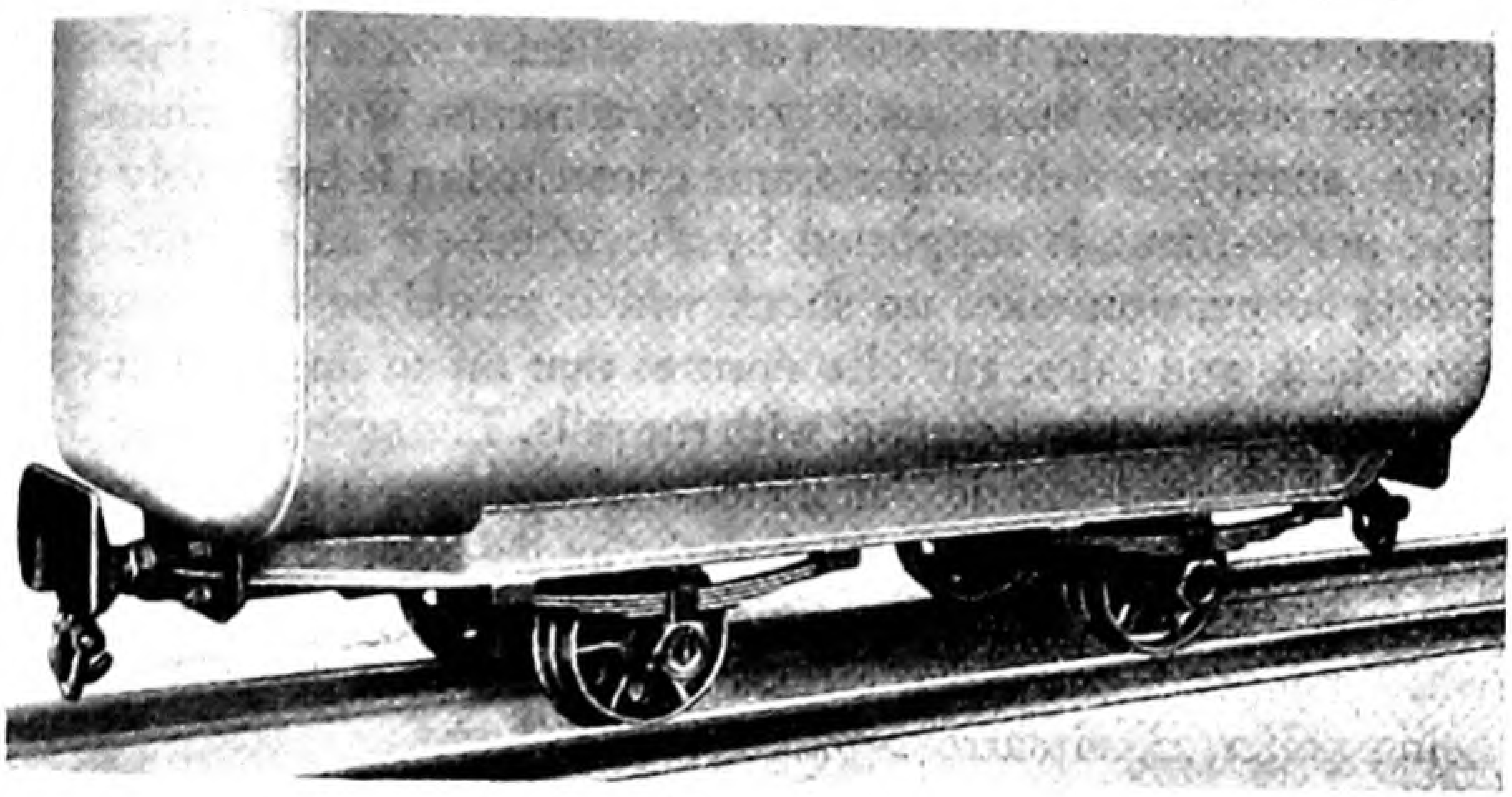


FIG 335.

cars are used together, so that there is no hindrance between cars when running over swinging platforms or other inclined planes.

The wheel-sets on both medium and large cars are fitted with taper roller bearings. The wheel-base and track gauge are dependent on stability and the ease of circulation through curves. The following table gives these particulars for several medium- and large-capacity mine cars. These calculated values may be reduced slightly in practice.

Capacity	*Wheel-base*	*Track Gauge*	*Radius of Curves to be Passed*	
			By Individual Cars	*By Trains*
litres	*mm.*	*mm.*	*mm.*	*mm.*
1,600	600	1,000	6,500	10,000
2,000	600	1,000	6,500	10,000
2,500	600/750	1,250	7,000	12,000
3,000	600/750	1,250	8,000	15,000
4,000	600/750	1,500	10,000	20,000
5,000	750/900	1,750	12,000	25,000

The wheel diameter of the standard German cars is usually 350 mm. (14 inches). The tendency was formerly towards a larger-diameter wheel. Repeated tests have indicated that the rolling resistance decreases with increasing wheel diameter. On the other hand, however, wheels with a diameter of 350 mm. have given consistently good performance with regard to travelling through curves, resistance to motion and wear and tear. The standardisation of wheel diameter has allowed similar standards to be laid down for brakes, tipplers, etc. The width of the wheel-tread is often increased in order to reduce the possibility of derailments. With medium- and large-capacity cars the under-frame construction is invariably used. The under-frame is supported by the wheel-set on springs. This spring suspension takes up shock which would be transmitted to the wheels and axles, and also ensures that on an uneven track the wheels will have close contact with the rails. The suspension springs are usually semi-elliptic, laminated-leaf springs. The suspensions may be provided outside and below the under-frame with the wheels inside and below the frame or vice versa. The arrangement chosen will be dependent upon the track gauge, the outside arrangement being preferred on narrow-gauge track.

Cars with bogies have been developed in which the bogies are also provided with spring suspension. The advantage of bogie construction is in the improved travelling through curves.

While in many cases medium-capacity cars are provided with fixed buffers, the large-capacity cars must be equipped with spring buffing gear. Generally, a central buffer is sufficient, but if the track curves are narrow and of small radius, it is better to enlarge the buffers and provide separate buffers on each side of the car.

For medium mine cars, two buffing-cone springs per buffer with a total deflection of 38 mm. ($1\frac{1}{2}$ inches) are suitable. In the case of a car of 3,000 litres capacity, either laminated leaf or ring springs may be used, while with larger mine cars, ring springs would be used only with a deflection up to 60 mm. (2·4 inches).

The couplings used may be similar to those on the smaller cars. The disadvantage of this type is the greater freedom and rigidity. The simple link coupling has to sustain very heavy tensile loads, especially when starting a set, and high shearing forces are introduced on the coupling when the train is pushed together. Handling becomes more difficult as the width of the car increases. For these

reasons, large cars have been fitted with fully automatic and semi-automatic buffing gear. Because of their elasticity, the draw gear can be used as buffers also. Semi-automatic couplings have the advantage of lower cost, simple construction with less play and easy handling. They also offer the possibility to connect large and small cars without loose intermediate links, using the normal link coupling. In general, a certain portion of the rolling stock consists of small and special cars for the transport of men and material.

Couplings have been developed in which uncoupling is not required in order to tipple mine cars. These swivel couplings are constructed so that individual or several cars can be turned on their longitudinal axis while the whole train is coupled. The main advantage lies in the saving of operators for coupling and uncoupling where skip winding is installed. The objections to this type are the difficulty in handling these couplings, if necessary, with high and wide cars, and the increase required in the diameter of the tippler. With high cars the coupling must be set very low and, in consequence, the tippler diameter is increased. With a car of 3 cu.m. (106 cubic feet) capacity, the tippler needs to be 3·5 m. (11 feet 6 inches) diameter instead of 2·2 m. (7 feet 3 inches) because the tippler axis must coincide with the axis of the couplings. This increase in tippler diameter means a correspondingly heavier construction and increase in power required. When decking mine cars into the tippler, the whole train must be accelerated and stopped each time, thus imposing a heavy strain on the chain creeper. For these reasons the swivel coupling has not found great favour in Germany.

In modern British mining practice, the National Coal Board have approved the recommendations of the Mine Car Committee and issued a standard specification for solid-bottom mine cars (SBMC. 1949), Standard Specification P.D. No. MM (49) 1. The Committee recommended the following standard range for these mine cars for use wherever possible in new mines and for reconstruction schemes.

The $3\frac{1}{2}$-ton car 'L' was included in the range because of the demand for such a mine car shown in a survey carried out by the Committee. The standard specification refers to the wide variation in density and size of run of mine coal and agrees that the capacity of the cars shall be stated as the 'water level' capacity of the body in

Car References	Capacity in cu. ft.	Nominal Pay Load	Recom-mended Rail Gauges	Width of Body	Depth of Body	Length of Body	Total Height above Rail	Length over Buffers	Wheel Base
		tons	ft. in.	ft. in.	ft. in.	ft. in.	ft. in.	ft. in.	ft. in.
A 67/ (Gauge) .	67	1½	2 0	3 0	3 2	7 3	4 6	8 0	3 0
B 62/ (Gauge) .	62		2 0	3 3	3 2	6 3	4 6	7 0	2 6
C 61/ (Gauge) .	61		2 6	3 9	3 2	5 3	4 6	6 0	2 3
D 85/ (Gauge) .	85	2	2 0	3 0	3 2	9 3	4 6	10 0	4 0
E 82/ (Gauge) .	82		2 0	3 3	3 2	8 3	4 6	9 0	3 6
F 83/ (Gauge) .	83		2 6	3 9	3 2	7 3	4 6	8 0	3 0
G110/ (Gauge) .	110	2½	2 0	3 6	3 2	10 3	4 6	11 0	4 6
H106/ (Gauge) .	106		2 6	3 9	3 2	9 3	4 6	10 0	4 0
I101/ (Gauge) .	101		3 0	4 0	3 2	8 3	4 6	9 0	3 6
J129/ (Gauge) .	129	3	2 6	3 9	3 2	11 3	4 6	12 0	5 0
K126/ (Gauge) .	126		3 0	4 0	3 2	10 3	4 6	11 0	4 6
L142/ (Gauge) .	142	3½	3 0	4 0	3 2	11 3	4 6	12 0	5 0

cubic feet. For the rough calculation of the pay load, 42 cubic feet of coal per ton is used as a basis and 24 cubic feet of rock or stone.

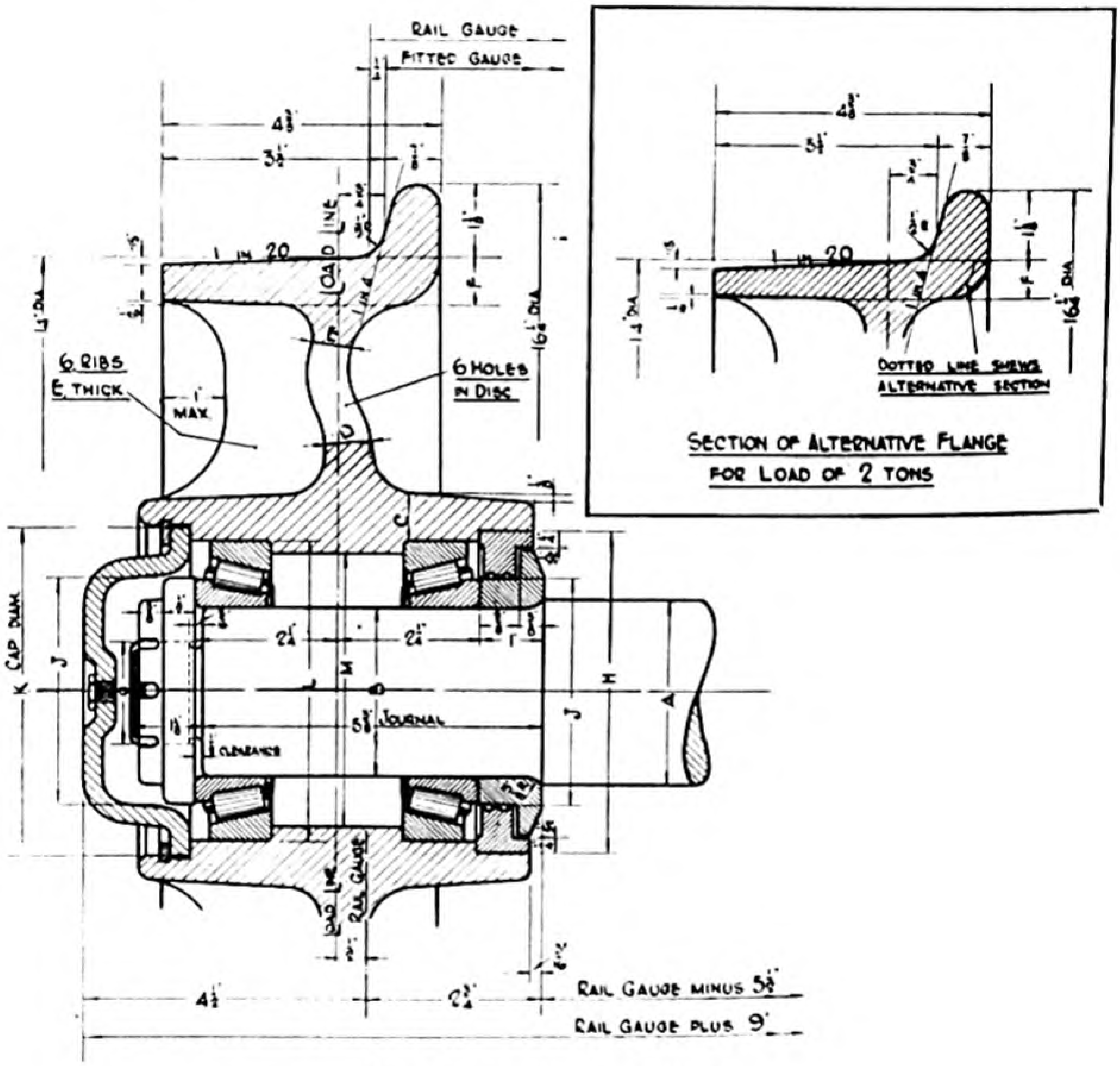
The length over head stock of the under-frame should be as great as possible and vary in steps of 3 inches where necessary to suit couplings. The body is specified to be made from $\frac{3}{16}$-inch to $\frac{1}{4}$-inch thick plate and riveted or welded. In the latter case, continuous welding internally is demanded. The body interior should be smooth and free from corrugations, wheel hoods, and well bottoms, to discourage the accumulation of dirt, and have a minimum radius of $4\frac{1}{2}$ inches in all corners. The inside or outside top edge should be provided with an unequal angle ring or other suitable section for stiffening purposes. Where further stiffening members are necessary, the smooth lines of the interior should be preserved. To facilitate discharge, the internal stiffeners, if any, must be sloped at not more than 35 degrees to the vertical. With regard to the under-frame design, the standard specification demands the use of 6 inches by 3 inches by 12·41 pounds per foot R.S. channels to B.S.S. No. 104 for cars from 61 to 85 cubic feet capacity, and 6 inches by $3\frac{1}{2}$ inches by 16·48 pounds per foot R.S. channels to B.S.S. No. 106 for the larger cars above 85 cubic feet and including 142 cubic feet capacity.

The wheels specified are standardised to three fitted gauges; namely 1 foot $11\frac{1}{2}$ inches, 2 feet $5\frac{1}{2}$ inches and 2 feet $11\frac{1}{2}$ inches to suit 2 feet, 2 feet 6 inches and 3 feet rail gauges respectively. The wheel diameter has been fixed at 14 inches and five standard loose wheels are specified complete with tapered roller, parallel roller, or ball and roller bearings, which are considered to meet all requirements. The wheel-tread may be $3\frac{1}{2}$ inches wide, to the profile shown in Figs. 336A and B.

Spoked wheels should have curved spokes, while the wheel-tread should have a smooth surface. The wheels should be of steel, hardened on the tread at option, or manganese steel, with the recommendation that a mild-steel hub should be cast in for machining. The roller bearings are disposed over equal distances on either side of the load line, while the dimensions over the hub caps when mounted on the axle should not be more than 9 inches wider than the rail gauge, with the distance between the wheel hubs $5\frac{1}{2}$ inches narrower than the rail gauge. The wheel-base is recommended to be not less than 40 per cent. of the body length and vary

in steps of 3 inches for each 6-inch variation in length over the buffers.

The height to the centre line of the coupling with 14-inch-diameter wheels should be about $12\frac{3}{4}$ inches (unladen). The composite buffing and draw-gear to be fitted with steel springs or rubber and steel disc packs. The draw-bar and buffer loads should



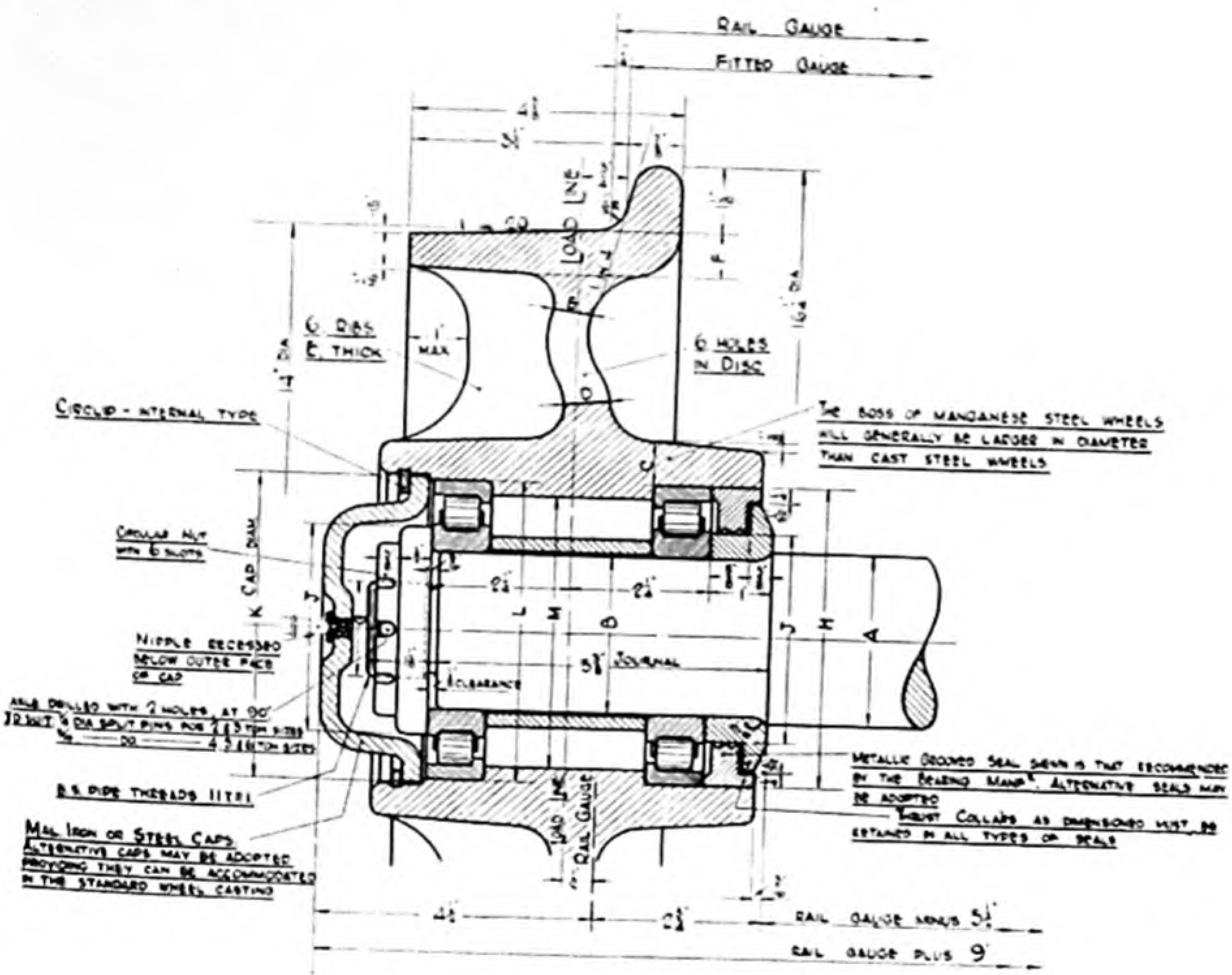
Taper roller-bearing wheel.

FIG. 336A

be transmitted by shear members to the longitudinal sole bars of the under-frame. The specification recommends the use of springs of the semi-elliptic, laminated-leaf type, because of their self-damping characteristics. Provision should be made in mine-car design for the fitting of brakes if required to all cars in the range. The brakes should be of the simplest type for parking purposes only and

capable of operation from either side of the car. The minimum requirement is a simple band or slipper brake upon the wheel at each end of the car.

Retarder engagement should be made on the under-frame and not on the axle. Standardisation of fittings for the several duties of engagement brackets for tipplers, retarders, creepers and rams is recommended.



Parallel roller-bearing wheel.

FIG. 336B

The specification draws attention to the divergence of opinion among mining engineers that the use of automatic couplings will reduce labour underground to any appreciable extent. Their need may be less with skip winding than with cage winding, and experience is still limited on their use in British mines. With cage-winding installations, difficulties may arise in ramming and traversing the pit bank, although these may be overcome with certain types of coupling.

With regard to orthodox couplings, the many years of experience

of the link and pin, hook and shackle, etc., on railways and in mines prove them to be reliable. The couplings are easy to uncouple safely but difficult to arrange for safe coupling, while slack is unavoidable. The use of loosely hanging links is precluded due to the possibility of fouling track fittings, creepers and stops, etc.

This specification is a considered cross-section of the opinions expressed by British mining engineers, and illustrates in comparison with the German practice previously described that a great deal of agreement is present on general and detailed aspects of mine-car design. The general arrangement of the British N.C.B. standard mine car is shown in Fig. 337A, which also illustrates the alternative arrangement for engagement brackets. The standard taper roller- and parallel roller-bearing wheels specified are illustrated in Figs. 336A and B, and one type of leaf-spring arrangement in Fig. 337B.

(*d*) *Appreciation of rail gauge required.* In order to decide on the particular rail gauge which should be used, the following factors should be considered. The advantages of broad rail gauges are:

(1) Greater stability in running and less lateral oscillation of the upper edge of the car sides when running on an irregularly levelled track.

(2) Easier travelling through curves of smaller radius and the possibility of choosing a longer wheel base. Travelling through curves will be better, the more the rectangle between the four wheels approaches a square. (N.C.B. standard—wheel base not to be less than 40 per cent. of length of body.)

(3) It is possible to install more powerful locomotives and obtain a better performance without increasing the height and the length of the locomotive, which would decrease the stability in operation and manœuvrability on curves.

The disadvantages of broad rail gauges are:

(1) The higher installation cost of ballast, sleepers and rails.

(2) The higher maintenance cost, since broad gauge track is affected to a greater extent by strata movement.

In the light of experience, the advantages claimed for broad rail gauge are to a great extent balanced out by the disadvantages. It is German experience that a rail gauge of 600 mm. (2 feet) is sufficient for large mine cars having a capacity from 100 to 140 cubic feet. The results of these investigations also showed that cars up to a

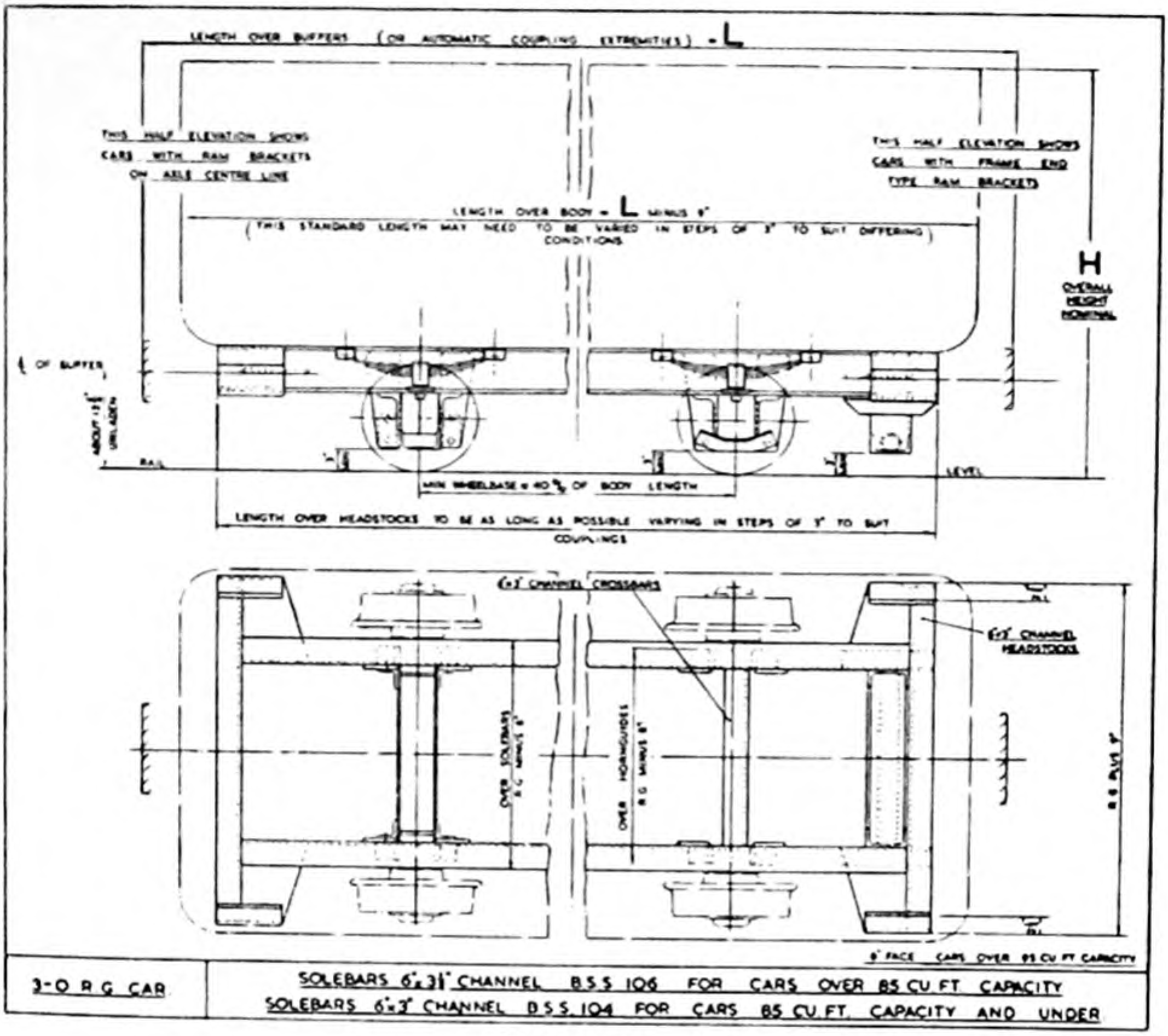


Fig. 337a

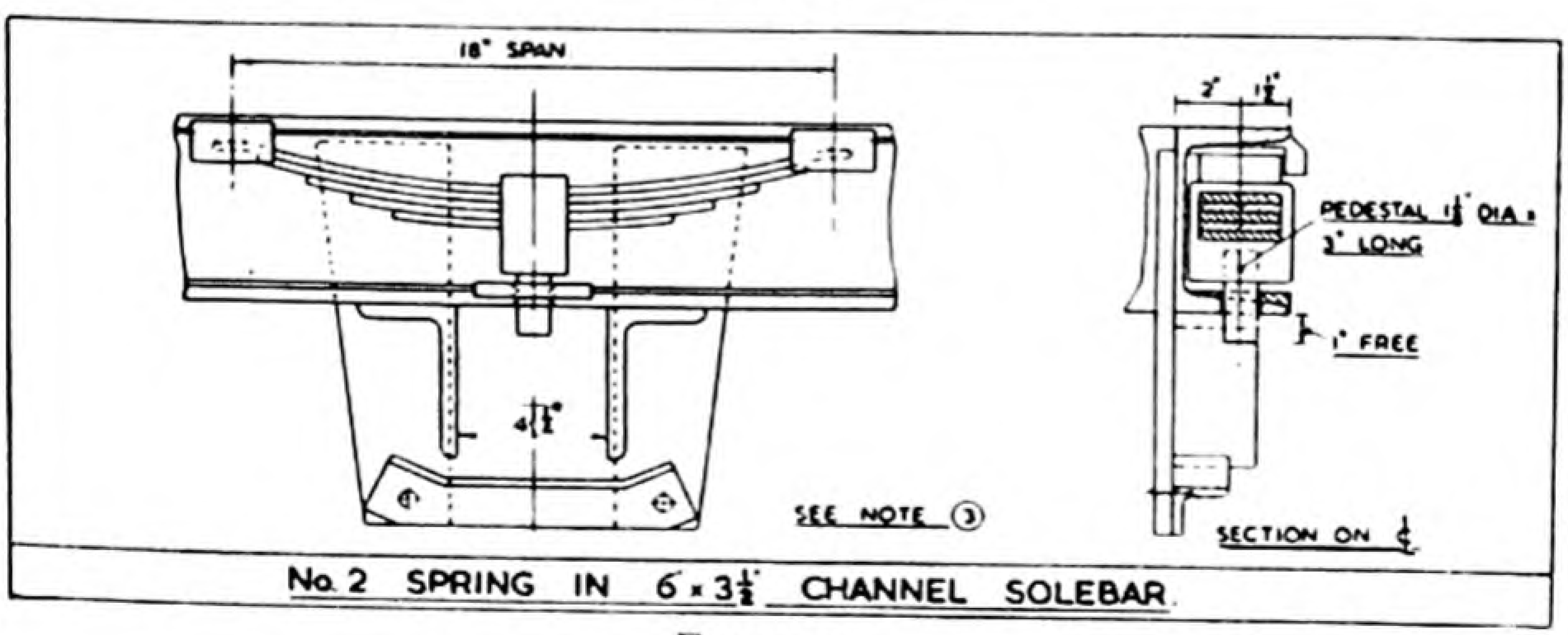


Fig. 337b

width of 1,500 mm. (5 feet) can travel safely on a rail gauge of 2 feet. This conclusion has the advantage of allowing transfer from large to small mine cars because the rail gauge can be retained in many cases.

On the Ruhr, the common rail gauge for small mine cars is from 1 foot 8 inches to 2 feet, while for medium and large mine cars up to 4,000 litres capacity (140 cubic feet) the common rail gauge is 2 feet. Where mine cars of over 4,000 litres capacity are installed, the rail gauge is increased to at least 750 mm. (2 feet 6 inches). It is considered that the quality of the foundation or ballasting of the track and the weight of rail employed are more important factors than the choice of rail gauge.

The N.C.B. standard mine-car specification for cars from 61 to 148 cubic feet capacity ($1\frac{1}{2}$ to $3\frac{1}{2}$ tons pay loads) are based on a range of 2 feet, 2 feet 6 inches and 3 feet rail gauges.

The comparison with American mining practice shows that the standard rail gauge is 3 feet 6 inches, but that under certain adverse conditions in low-roof mines the narrower gauge of 2 feet 6 inches is recommended. American haulage conditions differ appreciably from those experienced in Continental horizon-mining practice, since much heavier gradient work is tackled by trolley-wire locomotives than would appear to be permissible in British mines. The American hauls are in many cases very long, requiring a high pay load and the use of a powerful locomotive employing some 300 h.p. on a locomotive weighing up to 20 tons.

A British diesel-locomotive designer has emphasised that with low-built mining equipment, provided that the super-elevation on a curve is carefully maintained, derailment due to lateral thrust on the outer rail will probably take place long before overturning would occur, and that a sound argument can be made for a narrow rail gauge. The overturning speed of a 100-h.p. mine diesel locomotive with a 3-foot gauge on a 160-foot curve with 3-inch super-elevation of the outer rail is 53 m.p.h. This is a good example to show that locomotives are far more likely to jump the track than overturn. The divergent views of mining engineers on the choice of rail gauge has been summed up in an article in the *King's College Mining Bulletin*,* which gives the following conclusions:

Observations

(*a*) The only conclusion one can come to is that the adoption of different gauges has largely been a matter of individual inclination, and there is no clear evidence indicating what the choice should be for a standard mine gauge under certain circumstances.

* *King's College Mining Bulletin*, Vol. 4, Bull. No. 3, Haulage No. 7, 'Appreciation of Mine Track Gauge'.

(*b*) It does seem indicated, however, that where one is intending long hauls with possibility of heavy gradients, and trolley locomotives come into the picture, the gauge should be as wide as possible, and certainly not less than 3 feet.

(*c*) Although the evidence is not clear, there is some indication that a wider gauge gives more stability, and in view of the fact that track maintenance may not be so easy as on the surface, there is an element of weight to be given to a wider gauge.

(*d*) If there is a possibility of using drop-bottom mine cars, there is an additional weight for adopting not less than 3-foot gauge. The gauge may, of course, be limited by the width and type of mine car allowable by reason of cage dimensions and other reasons.

(*e*) The possibility of 'nosing' must be watched, but in general, the usual relationship of gauge to wheel-base would allow of either 3-foot or 3-foot 6-inch gauge with the larger mine cars (3 to 4 tons) or locomotives.

(*f*) In general, where change-over is involved the wider the gauge the simpler the track laying and actual change-over.

(*g*) There is no doubt that American ideas are influenced by the practice of intermittent loading with large cars on single track, or loop lines employing single track, whereas Continental ideas are influenced by the practice of continuous loading and double track, as well as provision for counter-flow of dirt for stowage. It is a point to be considered before deciding gauge, as to which is likely to be most efficient in the given circumstances.

(*h*) With regard to the implications of curve radius and size of junctions, it should be noted that while, theoretically, wide gauge means larger curves and junctions, the Americans appear to use speed control as an offset and work with remarkably small radius curves where economy suggests.

(*i*) The most significant remark is that of Schlöback in the Ruhr Report (Para. 4), who has experience of narrow gauge and suggests in effect that the present gauge should not be reproduced in new mines designed for large mine cars. This remark underlines the desirability of an unbiased approach to the problem, taking into account all the factors.

Conclusions

(*a*) Broad gauge, 3 feet 6 inches and 3 feet.

(1) Where long haulages are concerned, especially where the use of trolley-wire locomotives hauling heavy loads on heavy gradients or at high speed may be involved.

(2) Where the projected method of working allows single-track operation, e.g. projects which include intermittent loading, loop lines, the feeding of trains equipped with swivel couplings through tipplers by power, etc.

(3) In cases where flexibility is required to enable the use of drop-bottom mine cars.

(*b*) Narrow gauge, 2 feet 6 inches and 2 feet.

(1) Where large mine cars are to be used and the projected method of working involves a large amount of double-track working, e.g. on main roads and at loading-points, etc.

(2) Where the conditions in the working area are not suitable for the broad standard gauges and it is considered desirable to retain a common gauge throughout the mine. In this respect, main-line operation should not be subordinated to auxiliary-line operation, since efficient transfer arrangements for supplies, etc., are possible and the decision must be made taking all factors into consideration.

(*e*) *The choice of mine car required.* If the advantages and disadvantages of small and large mine cars are compared, the large mine car is characterised by several important advantages, which are as follows:

(1) Large mine cars have a more favourable tare-load ratio. The following table illustrates this factor with reference to several German mine cars:

Capacity	. litres	750	875	1,000	1,600	2,000	2,400	3,000	4,000	5,000
Pay load	. Kilos.	750	875	1,000	1,600	2,000	2,400	3,000	4,000	5,000
Tare weight	. Kilos.	535	616	720	850	950	1,050	1,520	1,600	1,840
Tare/Pay load .	.	0·71	0·70	0·72	0·53	0·47	0·44	0·50	0·40	0·37

This advantage is particularly significant, since it leads to a saving of power and thus to a reduction in cost both for vertical and horizontal transport.

(2) The consequent saving in locomotive power following a reduction in the tare-load ratio is still further increased by the lower rolling resistance of the large mine car due to the use of roller-bearing wheel-sets. Since fewer mine cars are required, the maintenance and supervision can be more effectively carried out.

(3) The number of mine cars in circulation is reduced, due to their larger capacity and the reduction in the circulation time required because of the higher speed of the haulage system. The reduction of the number of mine cars in circulation simplifies the time-table for the haulage system, while density of traffic is reduced and the system is more easily operated. The time required for coupling and shunting operations can be reduced, while loading and unloading is speedier with the larger cars. Experience has also indicated that better filling is achieved with the larger mine cars.

(4) The reduced number of mine cars and the favourable influence upon loading and tippling not only leads to a saving in manshifts but also results in a certain simplification of the equipment installed, both on the surface and underground. Thus, the number of tipplers is reduced and the installed capacity can be increased.

(5) The length of the trains can be reduced, and it is possible because of this to require smaller loading-points and shaft-bottom layouts. The limitation in the number of trains required to deal with the output may allow single-track operation in certain cases with a consequent reduction in the size of the roadway required.

(6) The construction of the larger mine cars, with their spring buffering, results in a more careful handling of the coal during transportation, although this advantage may be offset by the increased dumping height when tippling. The life of the rolling stock is, however, longer and maintenance costs are less.

(7) It follows that because of the increased stability of the large mine car, that the danger of accidents during haulage are reduced.

These advantages should be considered together with the following disadvantages:

(1) The large mine car cannot be universally used throughout the mine to the same extent as the smaller car, e.g. gate-road haulage will hardly be possible. For this reason, additional smaller cars must be kept, or expensive, special mine cars added to the rolling stock for many particular purposes.

(2) It will generally be necessary to drive longer and more strongly supported main roads.

(3) In individual cases, the size of roadways and staple shafts may have to be greater than the ventilation requirements demand.

(4) The proportion of mine cars necessary for reserve purposes, amounting to about 15 per cent. of the rolling stock for normal operation, is greater than that required for smaller cars, in which case 10 per cent. is an average requirement.

(5) The handling of larger mine cars by personnel is difficult and often impossible, while loading is also more difficult than with smaller cars. Increased mechanisation has, however, reduced the importance of this objection.

The initial capital cost for a given total required capacity is almost

the same for large or small mine cars. The reduction in the cost of large mine cars because of their lower tare/pay-load ratio, is almost offset by the higher costs of the additional refinements on the large cars. The initial capital cost, however, for a mine of a certain output will be somewhat less for an installation using large mine cars, since the saving effected by the quicker circulation of the traffic is generally greater than the extra capital cost involved due to the larger number of cars required to be held in reserve. The comparison of the operational costs is of greater importance in deciding on the type required. The operational cost per ton of output is from one-third to two-thirds of the cost using the smaller mine cars.

These conclusions can be illustrated by the following calculations, which are based on German practice, experience and costs.

Cost comparison between small and large mine cars

Initial capital cost

Daily output	4,000 tons
Annual output	1,000,000 tons
Weight of pay load per cu. ft.	0·0255 ton
Mine-car capacity required per cu. ft. of daily output (small cars)	0·7 cu. ft.
Mine-car capacity required per cu. ft. of daily output (large cars)	0·5 cu. ft.
Cost of rolling stock per cu. ft. of capacity	£1·42
Initial capital cost for small mine cars	£155,500
Initial capital cost for large mine cars	£111,100

Operating costs per ton of output

Item	*Small Mine Cars* *pence/ton*		*Large Mine Cars* *pence/ton*	
Depreciation	12·5% per annum	4·67	8·33% per annum	2·22
Interest	7% per annum	2·61	7% per annum	1·87
Maintenance	3% of capital cost per annum	1·12	2% of capital cost per annum	0·53
Lubrication	1·5% of capital cost per annum	0·56	0·4% of capital cost per annum	0·11
Total		8·96		4·73

The saving which can be made on the total operating costs is not shown in the previous calculation and should include the effect of the use of the large cars on the remainder of the working cost in accordance with the advantages given. It may be stated that, in

general, the use of large mine cars will inevitably lead to higher efficiency and greater economy in the whole of the haulage system.

The decision on the actual capacity of mine car to be installed will have to be based on the factors discussed, since the installation may be entirely new or the re-equipment of an existing system by changing from small to large mine cars.

In the case of a new installation it is important to consider whether skip or cage winding is to be used, since the shaft dimensions will have to be considered before the size of the car can be defined. With skip winding the advantages of large mine cars will exert a greater influence than with cage winding.

In level seams where the system to be installed will entail the mine cars running only on the main-road network—a policy which should be followed—and conveyor transport in cross-cuts and gate roads to the loading-point, the car size can be greater and roadway cross-section need not have the same influence on the decision of mine-car capacity. Where the ventilation requirements are already influencing choice of roadway cross-section, and where the strata pressure conditions do not preclude the maintenance of spacious roadways, mine cars of 120 cubic feet capacity (3 tons) can be generally recommended. If the installation is in semi-steep measures, where the cars must also travel in gate roads and be raised in staple shafts, cars of not more than 105 cubic feet ($2\frac{1}{2}$ tons) should be used. In the case where the rolling stock is to be changed over on existing levels, the difficulties of transition may be great or small, depending upon the extent to which the level has been used for car haulage previously and mine cars have been limited to main road traffic. It may be necessary, if the latter has not been the practice, to use large and small cars simultaneously, separated according to levels or task, i.e. haulage of coal or waste.

Where cage winding exists, the decision is influenced more decisively by the existing equipment than is the case where new plant is to be installed, and a compromise can usually be made. In the former case, the existing car size can be enlarged by doubling the length and increasing the height of the car. By these means, cars of medium capacity, from 65 to 90 cubic feet ($1\frac{1}{2}$ to 2 tons) were evolved.

It should be noted that many difficult and expensive reconstructions have been carried out with satisfactory results because of the

advantages involved. This argument has been applied to the change-over from cage to skip winding on many occasions, because of the other advantages associated with skip-winding practice. In the Ruhr, the use of large mine cars is increasing. The self-discharge mine car has not, however, found favour on the Continent in connection with horizon-mining systems. It is argued that the saving in rotary tipplers resulting from their use is offset by the considerably greater initial capital cost of the special cars and the maintenance of the discharge gear. The width of a self-discharge car is greater than the normal large-capacity mine car, and for this reason their use in cage winding is not recommended.

Where car traffic is required in gate roads in semi-steep and steep formations, the use of self-discharge cars for a shuttle-service on sections of the level may be practicable. In this case, the advantage of ease in emptying is important in order to reduce the cost of the recurring preparatory work for a large waste tippler station.

Section 8. *Main Loading-points*

The coal conveyed from the face can be loaded directly into mine cars running in the gate road, but in the majority of cases in level seams, conveyor gate-road haulage is adopted and the main loading-point is situated in the main road of the haulage horizon. Where the gate road is on the level of the haulage horizon, the loading-point will be situated generally at the junction of the gate road with the main haulage road. If the gate roads are above the main haulage horizon, i.e. between the latter and the return air horizon, a staple-shaft spiral chute connects the gate-road haulage to the main haulage level and the main loading-point will be at the staple-shaft bottom (Part II, Section 10). In the latter case, the loading-point will serve several faces and may handle outputs of more than 1,000 tons per day. In order to deal effectively with such large outputs, it is necessary to have a loading-point design which permits rapid mine-car loading and changing without spillage and the consequent formation of a dust-laden atmosphere.

An essential factor for the efficient operation of a main loading-point is the existence of a high-capacity main-road haulage system with the provision of double-track and separate traffic for full and empty mine cars. The transfer of discharge point into the main-road haulage will always be above the main-road track, in which case the

discharge height from either the gate road or intermediate road haulage should be a minimum to prevent unnecessary breakage. The gate-road conveyor can be graded upwards towards the loading-point and the roadway enlarged if necessary to provide adequate height. Where the coal is discharged from an intermediate conveyor leading from a staple shaft adjacent to the main haulage roadway, the same arrangement is made.

(*a*) *The layout of the loading-point.* The loading-point will usually be supported on steel joists set on brick pillars built along the roadway sides. The intermediate conveyor roadway between the haulage road and the staple shaft could be lined in the same way, leaving an adequate discharge opening. The space between the rails at the loading-point should be brick-lined so that spillage can be removed easily at frequent intervals, and guide rails should be provided to avoid derailments if the spillage becomes excessive before it is cleared.

The discharge-chute arrangement installed should allow a high rate of loading of the mine cars and reduce coal spillage during car-changing or caused by overloaded cars. The direction of loading in relation to the mine cars can be side-on or end-on, the latter method appearing to be more advantageous, since the speed of the train can be adapted better to the rate of feed from the chute and a greater density of loading is possible. A disadvantage, however, is the introduction of the change in direction of the discharge chute, the reduced clearance and the greater discharge height required.

The simplest form of side discharge chute incorporates a troughed sector door operated by a lever as shown in Fig. 338. With this form of discharge, bunkering is impossible. Since any stoppages at the loading-point due to peak loading inevitably affect the face operations, the provision of a loading-bunker with a capacity up to two car-loads is an advantage. The discharging of the bunker can be carried out, using a gate operated by a balance weight. The bunkers can be installed to load either side-on or end-on, as shown in Fig. 339, or to feed vertically. They can be introduced in the feeder arrangements for either gate road or sub-level haulage. A disadvantage of this method is the greater discharge height required and the additional excavation necessary. A loading-point of this type, however, will operate more satisfactorily and the dust formation will be less. The loading-point can also be provided with a vibrating feeder discharging on to a fixed delivery chute as in Fig. 340. The

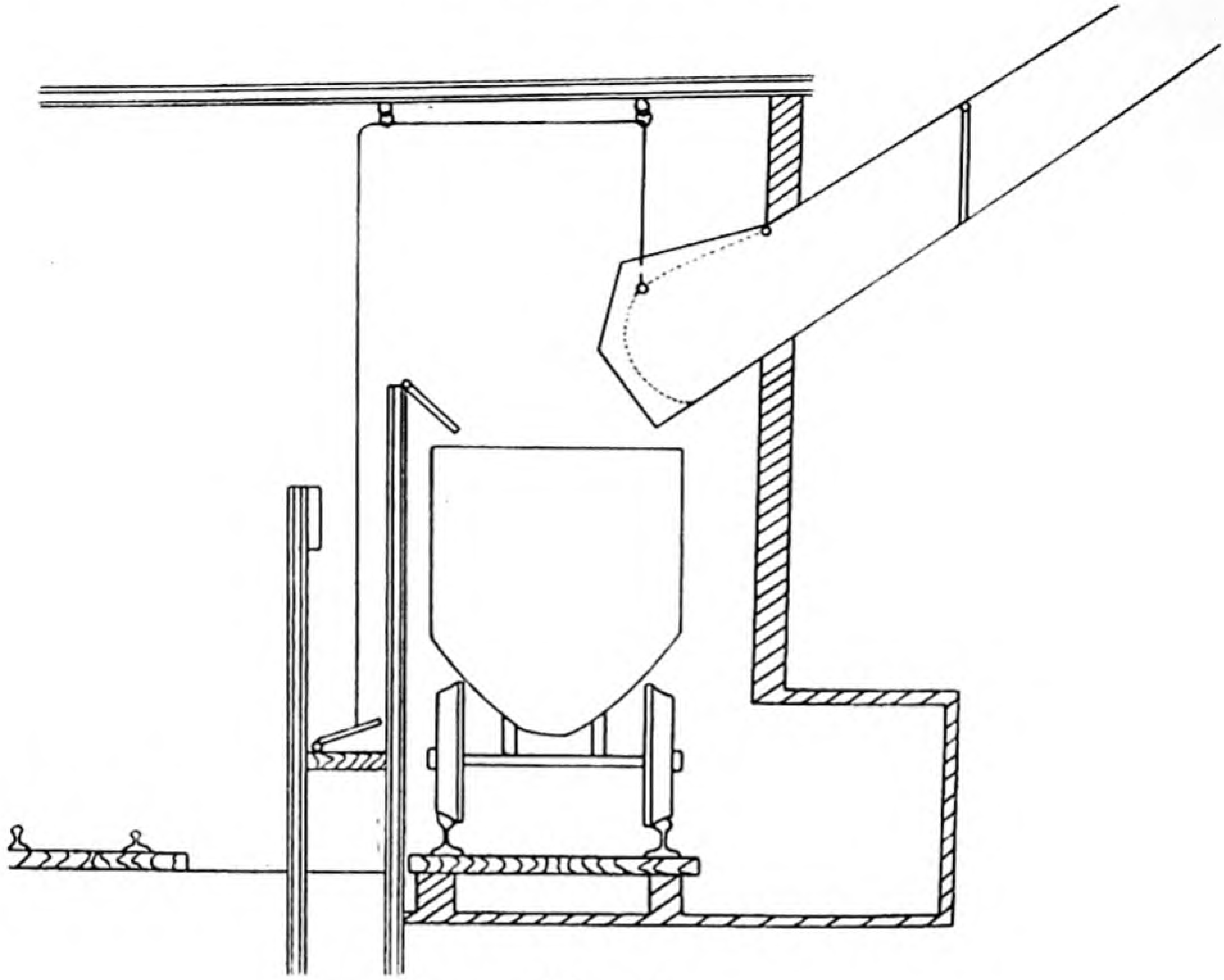


FIG. 338

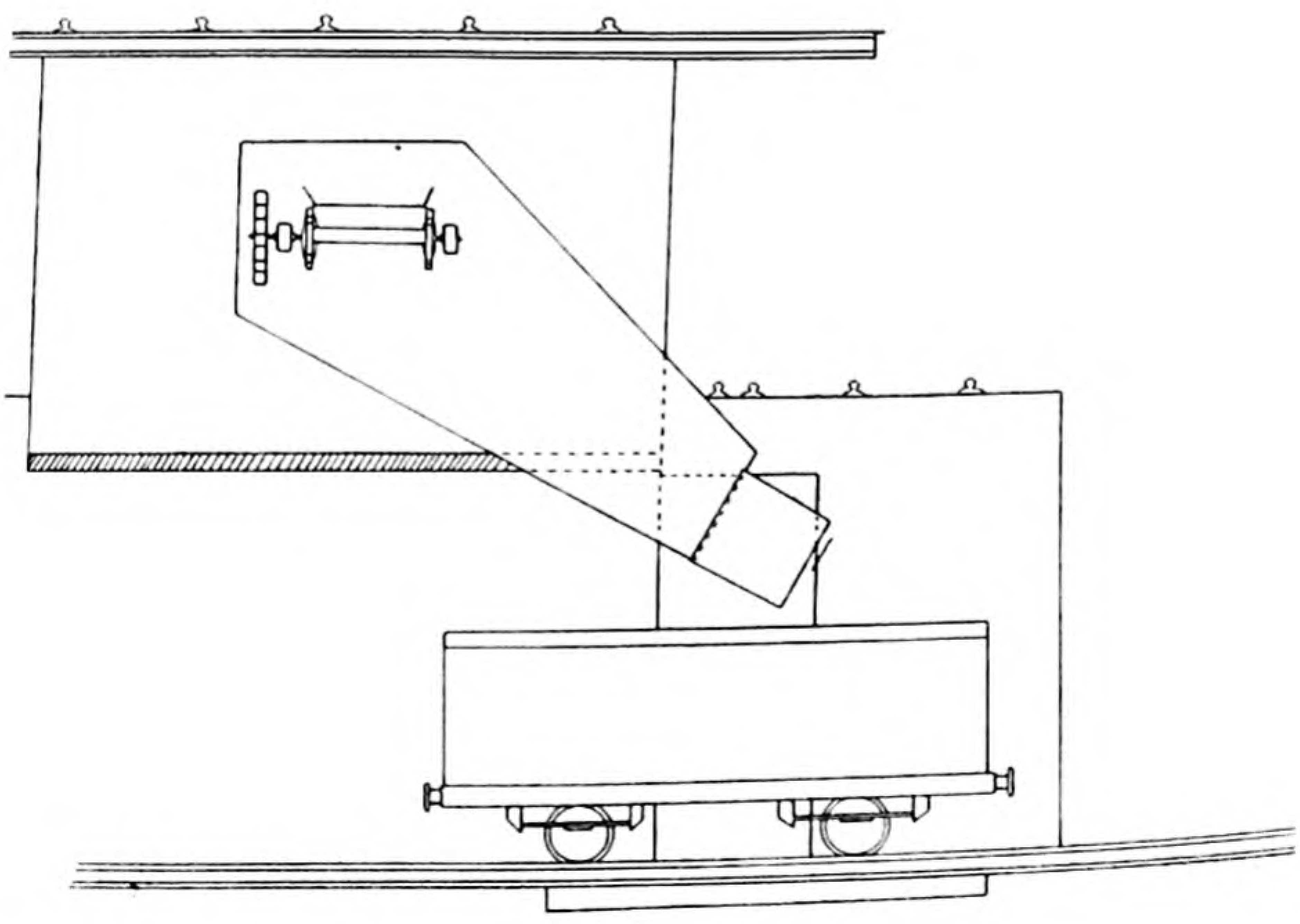


FIG. 339

feeder is a wide shallow-troughed conveyor, driven by a shaker-conveyor drive. In addition to the reduction in the discharge height required, a certain amount of bunker capacity and a uniform rate of feeding are obtained, while the discharge from the spiral chute is regulated. A disadvantage of this layout is the introduction of other

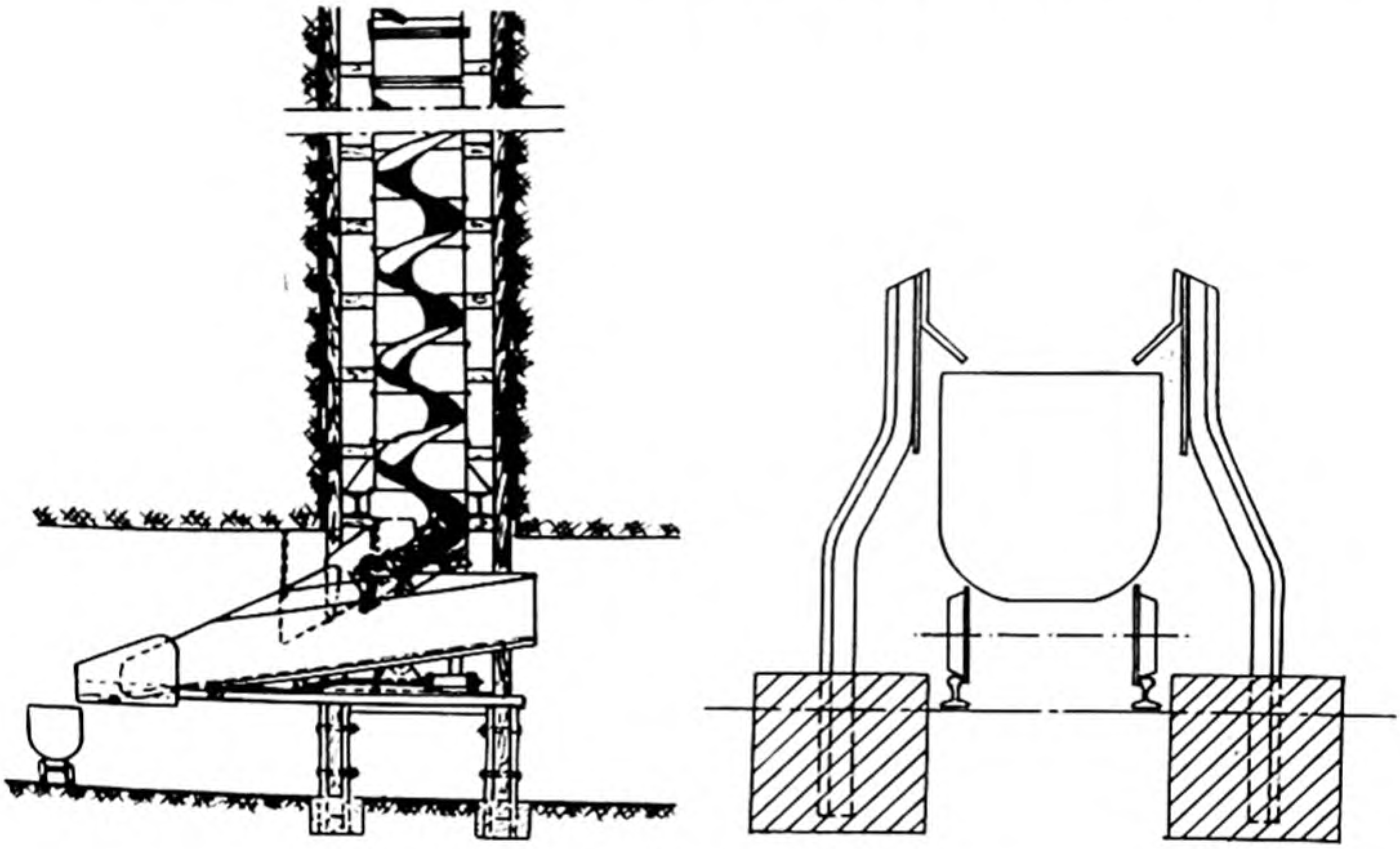


FIG. 340

mechanical equipment requiring additional maintenance, supervision and power consumption, and which can interrupt the continuity of the main haulage in the event of a breakdown.

Coal spillage may be reduced considerably if the discharge of coal can be interrupted during the changing of mine cars. At peak loading periods it may be impossible to do this, and the spaces between successive cars must be covered. For this purpose, a track ramp may be used, in which the adjacent cars below the loading-chute are tilted against each other in such a manner that the space between the car sides is closed during loading. Other methods adopted include covering plates or rubber aprons. The coal remaining on the covering plates after loading one car drops into the next car to be loaded during car-changing, and the plate is removed and replaced by hand between the next two mine cars. The same result is obtained using a

rubber apron which is fastened to the underside of the feeder chute. The apron projects into the car for a short distance and is held against the inside edge of the car by the accumulating coal. When the cars are moved forward, the coal lying on the apron is chopped into the following car. In spite of these expedients, coal spillage cannot be obviated completely due to the angle of repose of the coal and especially in overloaded cars. In order to reduce this trouble further, the loading-point can be provided with spill-plates, as shown in Fig. 340, which are fixed about 3 inches above the top of the mine car and

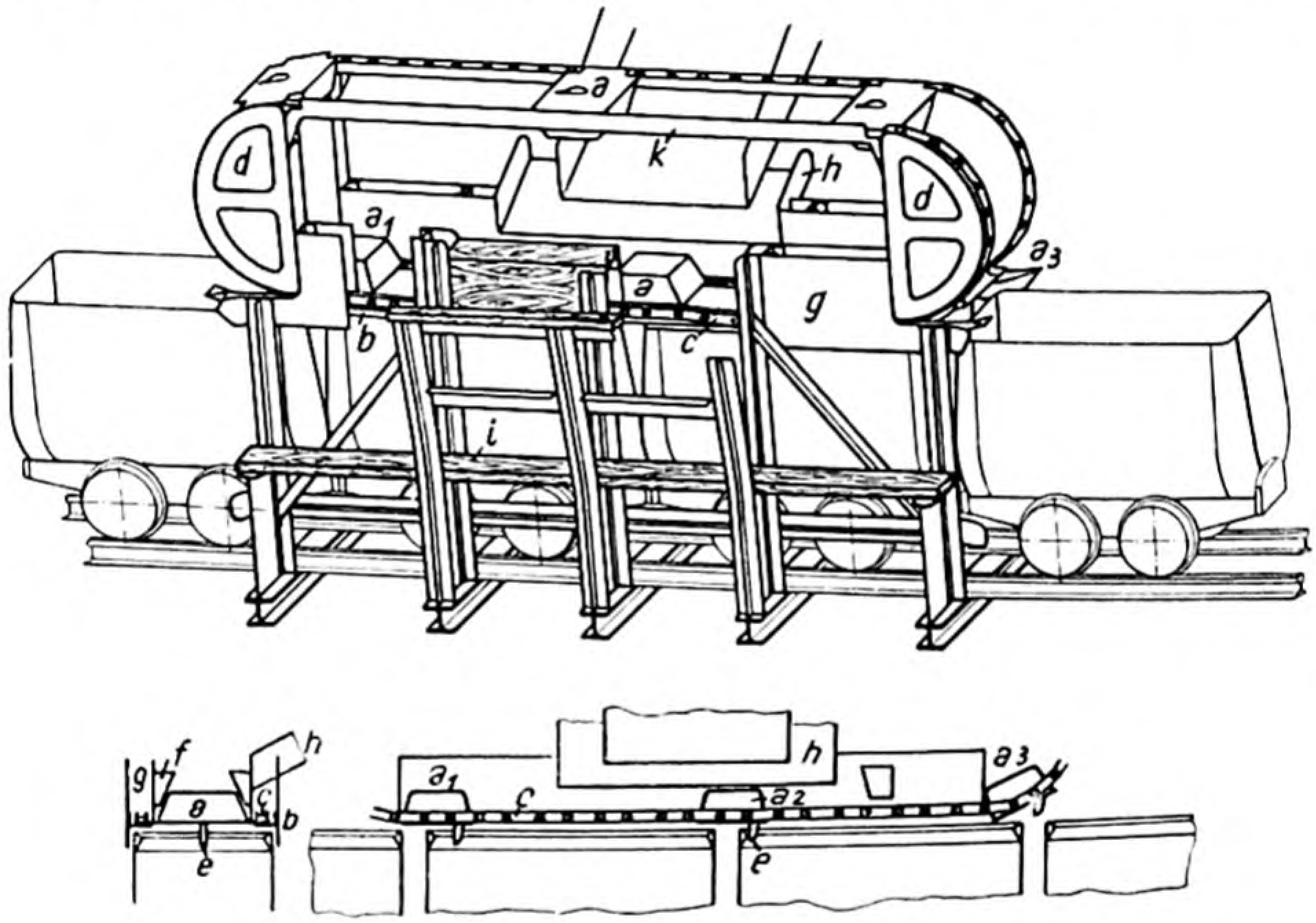


FIG. 341

project into the car covering the sides. With normal spillage, the well between the track can be cleared by hand, but with heavy peak loads and a greater output, the number of operators required to be engaged on this work may need to be increased. The loading-point can be equipped with a device incorporating rotating cover plates or boxes, in which the boxes are fitted to an endless chain running over two guide frames, as shown in Fig. 341. The boxes run on guide rails and sit over the space between the cars, with several inches of overhang on each tub. The boxes are moved automatically over the next space between cars as the train moves forward. Catch drivers attached to the covering boxes give a positive location for the boxes

and automatically accommodate the unit to the speed of the train. This apparatus has no mechanical drive and if side spill-plates are incorporated also, spillage is almost entirely eliminated and is independent of the skill of the operators at the loading-point. Since spillage can never be eliminated entirely and back-loading by hand is always necessary to some extent, the back-loading conveyor has been introduced, as shown in Fig. 342. The spillage drops through the track on to the scraper conveyor, which returns the coal into the empty cars behind the loading-point. The disadvantage of additional power-operated equipment still remains, together with the difficulty in emptying the pit by hand in the event of a breakdown.

(*b*) *Track layout and haulage.* The mechanical pushing of the mine cars is usually carried out on the empty side of the loading-point,

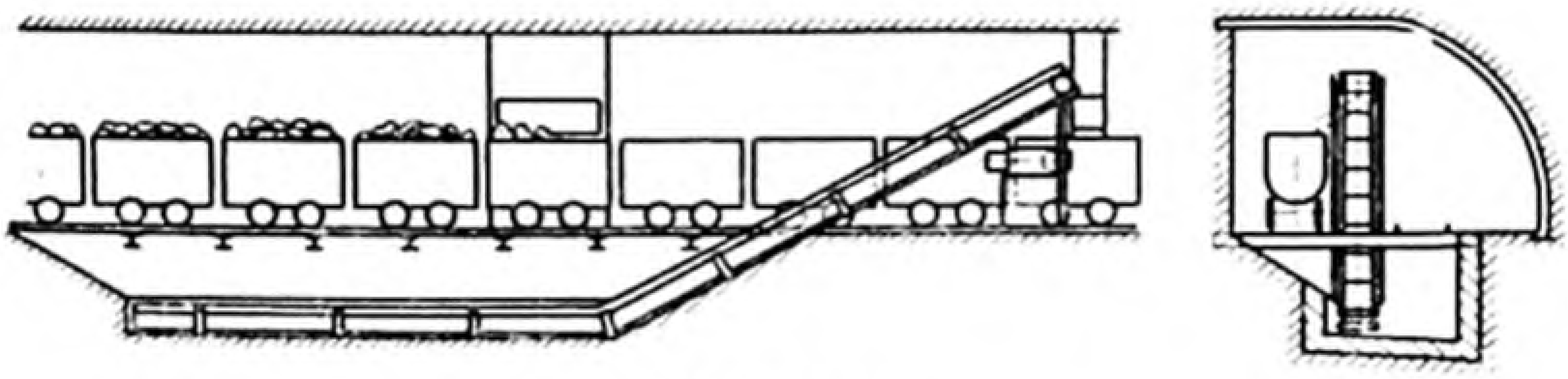


FIG. 342

using tandem pneumatic rams operating singly or in tandem, electric rams or a chain-creeper. A reserve ram should always be installed. The arrangement on the empty side of the loading-point is to reduce coal spillage. The chain-creeper is most expensive to install because of the chain-pit which is required, although the rams do not give the constant speed of advance necessary for an even loading-rate on the haulage. The rams are usually operated remotely with a hand or foot control.

The shunting arrangements commonly used include either a left-hand or right-hand switch, over which the complete train of mine cars is moved, either by the locomotive or a winch, from the empty to the full track. The average distance of the switch from the loading-point is from a half to three-quarters the length of a train, thus allowing a new train to be assembled. A second set of points is situated one train-length on the outbye side of the loading-point. Where the shunting is being done by locomotive, the train is hauled to the empty track, using a chain as in Fig. 343 (a), the locomotive returns and changes over to the front of the full train. In the case of winch

operation, the winch is usually located at the inbye set of points and the empty track is graded to eliminate the need for special rams. The new train gravitates to the loading-point and joins with the train being loaded.

In another shunting arrangement, the locomotive bringing in the empty train changes over to the full track before the loading-point and re-attaches to the rear of the train, which it pushes up to the

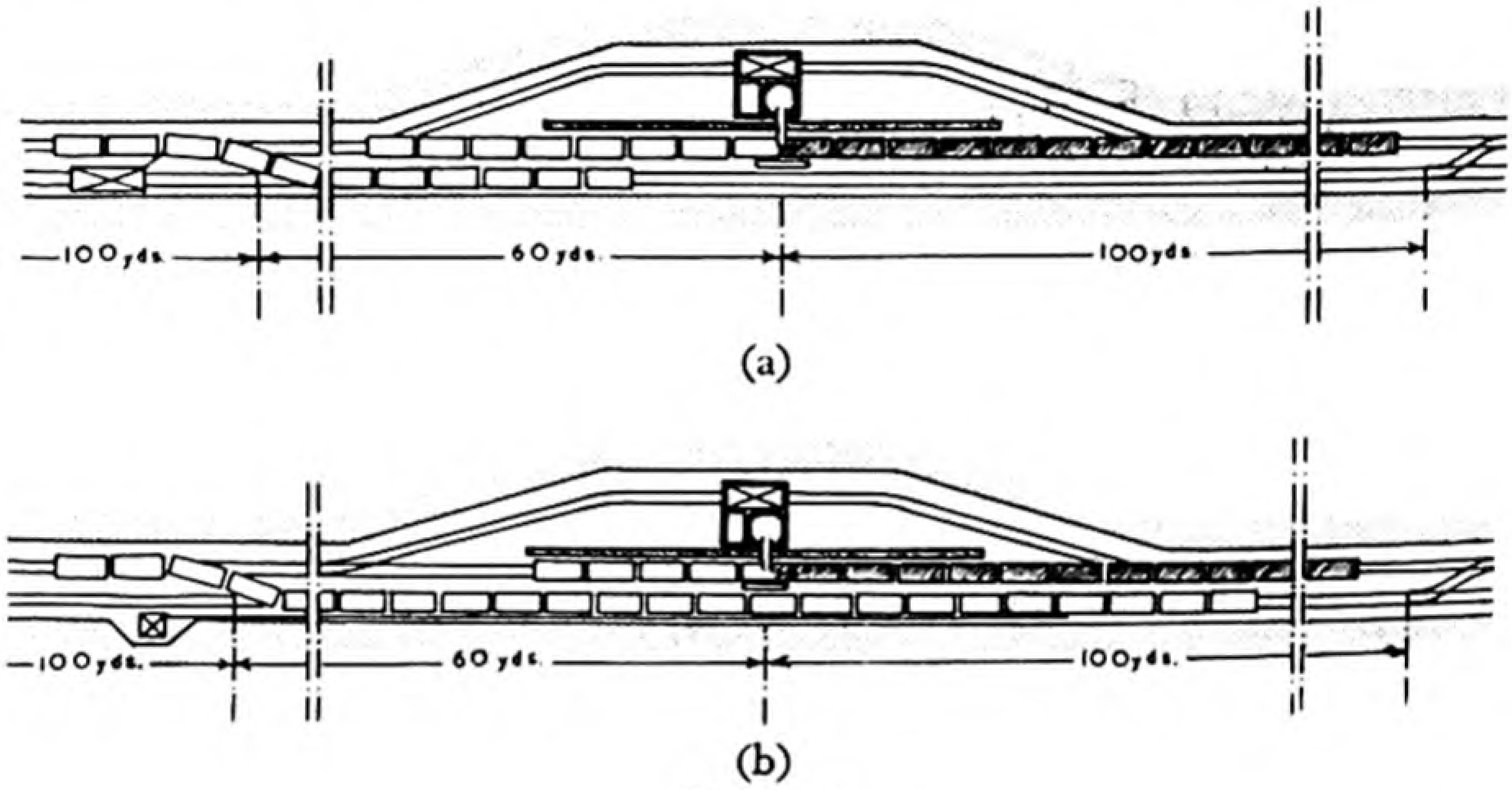


FIG. 343

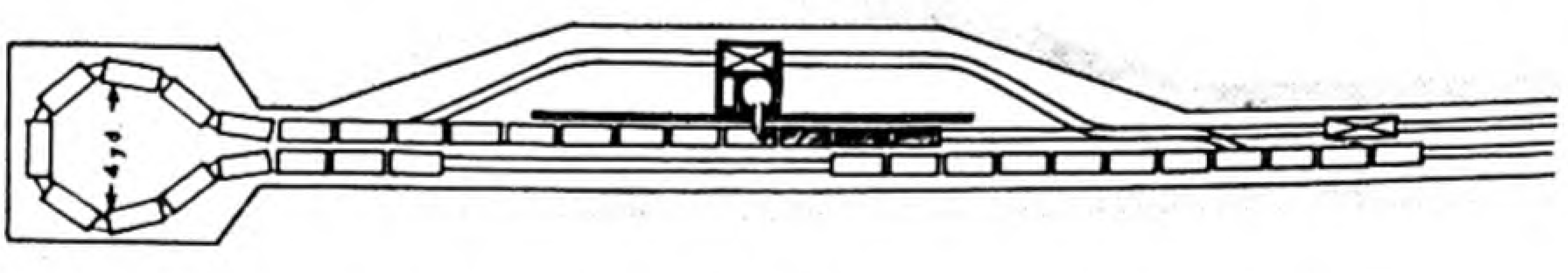


FIG. 344

winch. The locomotive is now ready to start out with the loaded train and is relieved of shunting duties, contributing to a better train succession.

Where several loading-points are situated in a cross-cut, it may be advisable to replace the hand-operated points with spring-tongue switches to increase the rate of haulage.

A further possibility is the use of the train turn-table or 'Karussel' shown in Fig. 344, round which the empty train is propelled by the ram at the loading-point from the empty to the full track. The operation of this layout requires the end of the preceding train to be still on the empty track, thus allowing the new train to be coupled

up. This turntable arrangement has an advantage when the loading-point is at the end of a panel, since complete trains can be shunted. The layout requires roadway enlargement, since the diameter is about 4 yards. The use of larger roadways will be possible only in solid ground. The method, however, eliminates the troublesome coupling and uncoupling of the mine cars which is necessary when roller switches or spring switches with a shunt-back are used, and has the advantage that no power is required.

Because the loading-point operator must be able to see the loading-chute continuously, he is exposed to the possibility of accidents by moving trains, etc. An elevated platform fitted with railings should be provided at the loading-point. The controls for the rams should be foot-operated and the chute door operated by a hand-lever above the platform.

To increase the loading density of the mine cars, i.e. the weight of coal loaded, pneumatic buffers are usually installed at the loading-point. The buffers or shakers consolidate the load in the mine car by concussions of the car sides or chassis. These devices are claimed to increase the charge of a mine car by up to 10 per cent. The loading-point should have signalling equipment installed.

(*c*) *General considerations regarding loading-points*. The type of loading-point installation described can deal with outputs of more than 500 tons per shift at 0·4 manshifts per 100 tons for loading, shunting and cleaning up. With an efficient main-road haulage system, however, providing a consistent flow of empty tubs, a loading-point with rotating cover-plate equipment can deal with outputs up to 1,500 tons per shift at from 0·15 to 0·25 manshifts per 100 tons. The capacity can be still further increased, using medium or large mine cars, giving a quicker train succession. The loading density will be increased and coal spillage reduced, resulting in a saving in manshifts.

PART IV

MAN-RIDING IN SHAFTS AND MAIN ROADWAYS

Section 1. General Introduction

Some distinction must be made between the provision and organisation of man-riding facilities in shafts, main roads in levels and in staple shafts. Man-riding in shafts is well known and has been carried

out for many years in general mining practice, but this does not apply to main-road and staple-shaft facilities. The introduction of man-riding is based upon the necessity to increase the productive time of underground personnel by reducing the time spent in travelling from the surface to their working place, and the conservation of human energy for the productive work to be carried out. The miner should reach his working-place as fresh as possible, and whereas the value of this is more difficult to assess than the time saved, it is equally important.

Section 2. Man-riding in Main Shafts

In general, each suitably equipped shaft (which connects the haulage or ventilation level with the surface) can be used for man-riding. The particular shafts which are chosen will depend upon their suitability, and due regard must be paid to the cost of the necessary equipment and its operation as well as to the effect on the reduction of travelling time underground.

In most cases the greatest proportion of the men are wound at the main winding shaft, which is the coal-drawing shaft. The surface and underground personnel are required at the main shaft, irrespective of man-riding facilities. The main shaft has the advantage that the locomotive haulage runs into the shaft bottom and can be used for the subsequent man-riding stage. The upcast shaft may also be used for man-riding, especially when the main shaft is overloaded or is only equipped for skip winding. Special decks may be provided for skips to enable man-riding to be carried out, but this increase in the weight of the skip requires larger ropes, and a skip affords less accommodation per trip than a cage. Skip winding is used only in special cases, and instead a second cage-winding engine operating in the same shaft or the upcast would be used. Where mines have a long lateral extension in their development, it is recommended that the ventilation shafts near the boundaries be equipped for man-riding to carry the men employed in such sections of the mine which are remote from the main shaft.

To increase the efficiency of man-riding, similar improvements are being investigated as for coal winding, (1) increasing the number of men wound in the same cage; (2) increasing the winding speed; (3) reduction in the intervals between winds.

The maximum number of men to be carried in one cage operated

on the Koepe system is limited to seventy according to German regulations. In individual cases the maximum number depends on the number of decks, the size of the standing area per deck and the height between decks. With a minimum height of 5 feet 9 inches, a standing space of about 2 square feet should be available for each man, this area increasing with reduction in height. Generally, the minimum height stated is exceeded with modern cages in which the upper deck has at least 6 feet clearance and the remaining decks at least 5 feet 9 inches.

In addition to the previous regulation for Koepe winding in Germany, another prescribes that the total load on the rope when used for man-riding should not be more than 90 per cent. of the total maximum load carried by the rope, a condition which is usually fulfilled. The most important restriction, however, is the provision of a safety factor of 9·5 for the winding rope used for winding men as against 7 when carrying coal and stone, these factors being referred to the aggregate breaking strength of the rope. With larger depths and according to the weight of the rope after 600 to 700 yards, the factor of safety will become decisive for the computation of the rope diameter required. Special regulations are being drafted for Koepe winding in Britain.

The time per wind will depend on the average speed of man-riding, which in turn depends on the maximum speed and the length of the acceleration and retardation periods. In Germany, the maximum speed is restricted to 10 metres per second for steam winding and to 12 metres per second for electric winding, because of its smoother operation and more effective means of control. The effective maximum speed will in all cases depend on the condition of the shaft and the rope.

The reduction in the waiting period between windings is important, the number being influenced by the frequency of decking. This time can be reduced considerably if all decks are entered and left simultaneously, this being achieved by the use of man-riding platforms on the surface and a man-riding pit or level below the coal-winding level at the shaft bottom. The installation of several platforms on the surface does not usually give rise to any difficulty. In the case of a four-deck cage, three platforms will be required. They are mounted by staircases constructed from steel sheet and are collapsible, being raised during the coal-winding operations and

lowered for man-riding. There is often a limitation on the space at the shaft bottom for the number of platforms required. Where a swinging platform is used for the decking of mine cars, a platform cannot be installed for the deck immediately below the level of the pit bottom. The provision of only one platform is more usual due to these limitations, and a cage with four decks needs to be shifted only once.

The reduction in the waiting periods can only be achieved if simultaneous entry and leaving of the cages is carried out. The men must enter the cage from one side and leave from the other, and the direction of movement of the men must be the same. Where entry and leaving is carried out on the same side, the pause increases by about 11 per cent. If fixed cage guides are being used, these should be collapsible at the man-riding platforms, so that the door of the cage is completely free for entry. Generally, decking is quicker during ascending than when descending.

Shortening of the signalling times can also be an influence. Modern shaft-signalling equipment combines the man-riding signals and materials-winding signals, which can be operated by the banksman at the man-riding platform and the onsetter in the shaft-bottom man-riding station.

Control of man-riding. The general feature of control in man-riding is the necessity to arrange that men from one face or panel ascend and descend simultaneously. If this is achieved, the men of one group will avoid waiting for the arrival of other men, and the net working time is increased. The simultaneous arrival of personnel and the stopping of work in large mechanised districts is especially important. The overman who descends with the men can previously start the distribution of work on the surface. The simultaneous winding of small groups or partnerships is equally important to avoid waiting-time. Where mechanised man-riding is available on the main haulage road, the same groups of workers as in shaft-riding should be carried at the same time.

This kind of organisation can be rendered difficult or made impossible if a great number of the miners reach the colliery by rail or bus, since some of the men will arrive after others.

Section 3. Locomotive Man-riding Haulage

Man-riding on locomotive-haulage roads is being widely extended. Compared with shaft winding, the train has a greater capacity

taking more than one cage load. The speed of driving, however, is less, the length of the travelling time is greater and the direction of travel is not consistent, so that simultaneous man-riding on the main-road haulage as in the shaft cannot be achieved.

The cars used are the usual mine cars, into which seating planks or comfortable seat-girdles are hung. Since the larger mine cars are

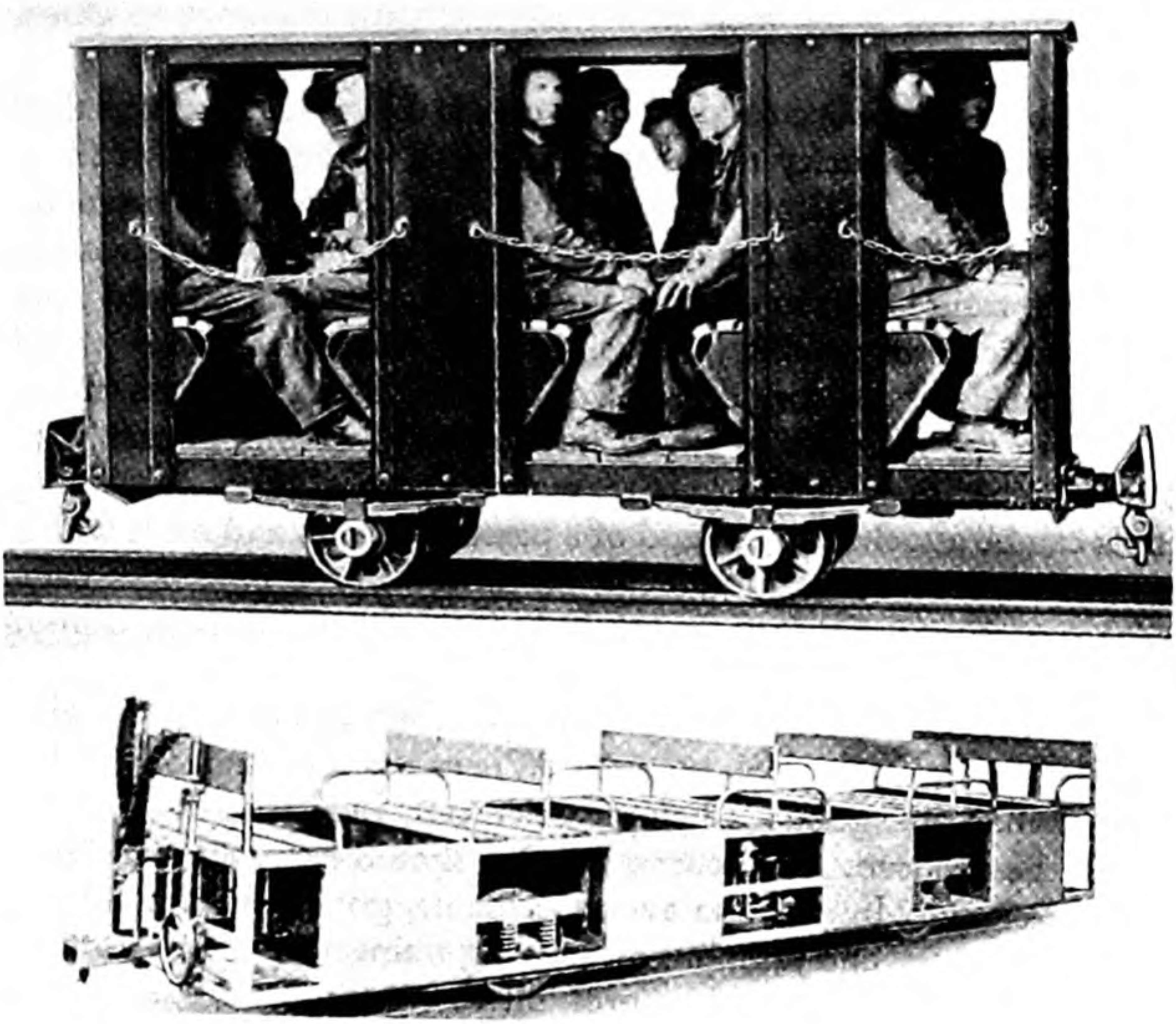


FIG. 345

sprung and have spring buffers, these are better than the smaller tubs. As a protection against too heavy shocks during braking, a 'drag' or wooden baton is often used on the wheels of the last car so that the wheels slide on the rails.

Special man-riding cars offer more comfort and different types have been designed, two of which are shown in Fig. 345. Most of these cars are covered to offer protection to the men against overhanging supports, projections and falling rock. Another common feature is

the fixed seats upon which the men sit astride or back to back, and in most cases the cars are sprung. These springs are either carriage springs or are under the seating planks. The springs are expensive, but reduce the danger of derailments. The length of the cars varies between 7 and 14 feet and they carry from 6 to 20 men. The sides of the cars are lower than those of the mine cars and are cut in a manner to facilitate changing of seats. The best form to obviate accidents is the type on which the men sit astride and with the sides to elbow height.

An advantage of special passenger cars is that with equal size they have a greater capacity than mine cars, and in the case of trolley-wire locomotive haulage, the provision of covered cars is essential. A disadvantage is the provision of special siding accommodation when the train is not in use and the fact that they can be used only at the particular travelling times, since their use at other times would cause considerable disorganisation of the normal coal-haulage traffic.

It is in the interests of safety not to carry persons in cars with other mine cars containing waste, material or coal. Cars containing tools can be attached to the end of a passenger train and tools which project over the sides should not be carried in the passenger cars. Where persons are being carried during a shift, mixed trains cannot be avoided and the passenger cars should be attached immediately behind the locomotive and should not be followed directly by cars containing timber or such material. Where the simultaneous transport of men and goods is being carried out, the speed of the train should be reduced. All passenger trains should carry special lights or light-signals. In order to avoid accidents, good discipline of the men is necessary and jumping off moving trains near the pit bottom or at inbye stations should be prohibited.

(*a*) *Man-riding stations.* A starting and arrival station is a definite requirement in order to maintain a time schedule and to secure the safety of personnel. This platform or station should be exactly fixed, marked and used. A station should be as near the shaft as possible so that the walking time from the shaft to the train is as short as possible. The inbye station should be so located as to limit the travelling distance to the face. It should be possible to enter and leave the cars either from or to the road to the working district. Trolley-wire locomotive haulage requires particular measures if special cars are not used. The roads along which the locomotive travels must not be

entered while the overhead line is alive. Track switches should be provided at the appointed stations. Where the station is not at the end of the line, the dead section of the overhead line should be bridged by a cable the same length and diameter as the overhead line. In addition, red and green lamps should be provided every 10 to 15 yards, which must be visible and show whether the overhead line is alive or dead.

(*b*) *The range of application of man-riding haulage.* The application of man-riding depends on the time which can be gained and the possible resting of the men for their real productive work. The cost of coal transport and its increasingly more difficult organisation play a major part in the decision to adopt man-riding facilities. The time saved is influenced by the driving speed of the trains, the waiting time and the distance to be covered, and in addition by the walking time of the men. The latter factor depends upon the height and general condition of the travelling road, as well as on the ventilation conditions. The driving speed of the train and the allowable walking time is fixed by the manager for each individual case.

It is generally agreed that, according to the importance of the above factors, a gain in time affecting the net working time will be realised by man-riding haulage in favourable cases beyond a distance of 1,000 yards. It is more general, however, to anticipate the benefit after distances of from 1,500 to 2,000 yards.

The advantages to be gained by the introduction of man-riding facilities outweigh the few disadvantages, which can be removed in any case by simple means and an appropriate organisation. Easy operation of man-riding haulage requires careful arrangement with the transport of loaded and empty coal trains.

Where man-riding has been arranged, all men who are using the same road must ride on trains as far as is possible and foot-travellers should be prohibited when locomotives are in use.

Where trolley-wire locomotives are used, the overhead line must be switched off at the time when men are travelling the same road. This arrangement often causes difficulty with the men who are working near to the shaft and do not require transport. The possible solution is to provide an alternative road or the train must start early enough so that the men from the district near the shaft can await its arrival at the shaft. The switching off can be carried out with a track switch or automatically by the passing train. The advantage of a

shorter driving time for the passenger train is sometimes nullified by the compulsory early termination of the coal traffic before the end of the shift. The degree of utilisation of the locomotives is increased by passenger transport. The necessity to add to the locomotives available for this purpose will depend on the capacity of the locomotive haulage and the efficiency of the organisation employed. It may often be recommended to hang on additional empty cars to passenger trains in order to gain the full benefit of the tractive effort available. Thus, the advantage is gained of bringing in empty trains to the loading station earlier in the shift.

Section 4. Man-riding in Staple Shafts

Man-riding facilities in staple shafts, even at small depths, is more preferable than travelling by foot, the limit of economy being about 15 yards. This low limit is due to the small speed of the man during climbing and weariness caused by the climb. With a depth of 50 yards the man reaches an average speed of 0·13 yard per second while ascending and 0·16 yard per second when descending. The man-riding speeds at the same depth would be from 2 to 6 yards per second. The ratio between man-riding and climbing times becomes even more unfavourable if the overall travelling time is considered, since a minimum distance between each man is required. The speed of climbing, overall, will depend on the slowest man, and with greater depths this factor becomes more important. It is essential that the winding engine installed in the staple shaft is adapted to the total number of persons to be wound, otherwise too long winding times will result, considerably reducing the advantage to be gained. Where coal transport in the staple shaft is by spiral chute the capacity of the single-cage system for materials will be based on the man-riding period and not on the drawing of materials.

Man-riding in staple shafts offers such advantages that the cost of additional safety devices need not be considered. The application and type of safety devices required depends upon the depth, winding speed and number of persons being carried at one time. With shafts up to a depth of 50 yards, a maximum speed of 6 feet per second and a load of 10 men is prescribed in the Ruhr. In this case, only overwind apparatus is required as an additional precaution. At greater depths, the speed, number of men carried and safety devices required are more strictly prescribed. Staple shafts, which according to their

purpose are equally important as main shafts, must be equipped and operated in a similar manner.

The hoist installed for man-riding must be equipped with over-winding equipment and over-speed control to limit the winding speed to a maximum of 12 feet per second. The normal polyphase induction motors are used with limit switches, and centrifugal over-wind control, over-speed and power switches as safety devices.

Special care must be taken with regard to the electric supply system. The commonest signalling apparatus is the simple rapper, which is operated by a wire and which strikes against a plate. In addition to the rapper which is used during coal-drawing and man-riding, a rapper is installed for signals in the opposite direction. These signalling systems must be arranged so that they can be operated from the cage from all places in the staple shaft, down to the lowest winding level. Electric signalling systems are being introduced into the larger staple shafts.

In order to avoid accidents by drowning in the event of the cage being lowered too far, a free space of at least 3 feet must be provided immediately below the lowest winding level. With shafts over 50 yards deep, this space should be more. The gliding beams must be either contracted or thickened at the shaft bottom in the same manner as in the main shaft. A corresponding free space is required at the upper landing, and for additional safety it is recommended that only the top deck be used for man-riding in a multi-deck cage. Where skip winding is in use, the question of adding man-riding decks to the skips has not the same importance in determining rope size, and the increased weight of the skip would not be a decisive factor.

The space requirements for man-riding at the bottom of a staple shaft can generally be fulfilled without any further excavation work. Entering and leaving the cage can only be done from the side on which the safety devices and the banksmen are located. This condition has not the same effect as in main-shaft winding, since simultaneous winding is not taking place.

Section 5. Organisation of Man-riding

The whole of the efforts of the organisation in the case of man-riding is towards the completion of all operations in the shortest possible time. Thus, not only has transportation to be carried out

with the greatest precision, but the time between the main shaft and the second stage of the transport system, and the time between the second stage and the third stage at the staple shaft, must be reduced to the lowest possible period. In addition to the technical efficiency of the winding and haulage installations and the number of men, the good will and interest of the men are of considerable importance in realising the purpose in full.

The proper division of the number of men into riding groups is important. The number and membership of these groups should be

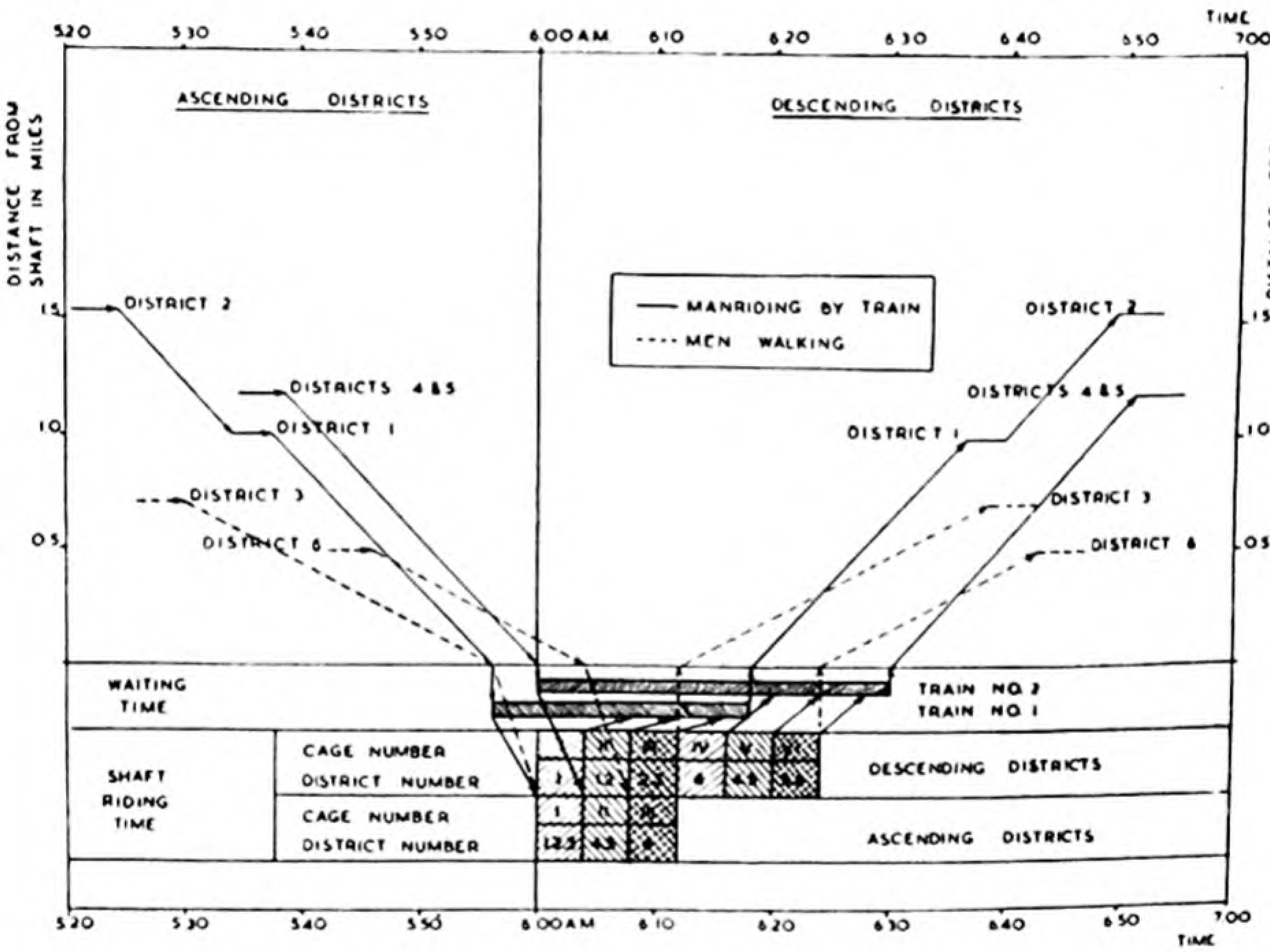


FIG. 346

allied to the membership of teams working in adjacent panels or districts as well as to the capacity of the winding installations to be used in sequence. These groups will ride together during descent or immediately after each other. On arrival at the shaft bottom, time should not be wasted and the train should start immediately. The travelling time should be examined to see if the admissible maximum speed is being utilised. The subsequent staple-shaft man-riding should be carried out in the manner and order found to be the most suitable. Man-riding in staple shafts for the same groups always begins at the same fixed time. The passenger train provided for these

groups should be at the fixed place at a definite time and the men should ascend at the main shaft immediately they arrive. Mechanised man-riding in the main roads immediately achieves its full value when man-riding in the main shaft has been well planned and vice versa. The secret in fixing the most rational system of travelling times and the manner in which travelling should be done is to give sufficient elasticity to the plan so that it can last for several months at least. The deviation from the travelling times and pauses should not be great, however, otherwise the objective cannot be reached.

The preparation of diagrams as shown in Fig. 346, based on time-study observations, has proved to be very valuable. In the example shown, the men from districts 1, 2 and 3 are descending in the first three cages. The total winding time is 12 minutes and after 6 more minutes the first passenger train starts with these men. At the same time, man-riding in the main shaft has continued with three more cages for the men for districts 4, 5 and 6, again over a winding time of 12 minutes. The men take passenger train 2, which starts 6 minutes after the arrival of the fourth cage. The diagram also shows the movements of the ascending men.

PART V

MAIN-SHAFT BOTTOM LAYOUT

Section 1. General Layout for Cage Winding

(*a*) *Storage capacity and length of sidings.* The pit or shaft bottom is the connecting link between the main-road haulage and the shaft-winding system. It must be capable of accommodating, smoothly and efficiently, the full trains arriving from the inbye districts, and of storing empty cars or dirt and material tubs which are gathered into sets or trains at the pit bottom.

Another important function of the pit-bottom arrangements is to prevent fluctuations in the main-road haulage from affecting the shaft-winding facilities or fluctuations in the shaft winding from affecting the smooth running of the main-road haulage. In this way it will be possible to use the road haulage to its highest capacity and, what is more important, to make full use of the capacity of the main-shaft winding equipment. Fluctuations in main-road haulage will be experienced usually at the beginning of a shift, when the supply of

coal is gradually increasing. This fluctuating supply is overcome if coal working is staggered from one or more faces during the night shift and the pit-bottom sidings are full at the commencement of the winding shift. During the course of the shift the trouble arising from day-to-day difficulties, either at the face or on the haulage, are compensated by the pit-bottom stocks. To absorb the fluctuations in the haulage system, the pit bottom must be equipped with a certain storage capacity. Under normal conditions it is usual to have both sides of the pit bottom filled up to half of their capacity with tubs. When the output from the main haulage has reached a peak, or there is a stoppage in the shaft-winding system, more tubs will be gathered on the full side and the empty stock will decrease. The reverse position occurs when the output of coal from the haulage system is decreasing and the winding system is working normally; in both cases the total number of empty and full tubs in the pit-bottom circuits is constant.

Assuming that the average capacity of the main-road haulage and shaft-winding system is the same and that fluctuations in one of the systems should be balanced as nearly as possible, the calculation of the required tub-storage capacity should be based on:

(1) the highest and lowest performance of the roadway haulage over a given period, and
(2) the shaft-winding time.

From these two factors it should be possible to find the change of balance in the full and empty tub distribution and to calculate the corresponding storage capacity. In practice, however, such calculations can only be rough estimates, since no reliable initial values are available and an estimate has to be made of how long the roadway-haulage system can continue to work in the event of a breakdown in the other system. Quite apart from this consideration, it is unavoidable in practice to make a compromise between the balancing of heavy fluctuations in the pit bottom and the disadvantages arising from the greater extension of the pit bottom and the larger number of tubs kept out of circulation.

Practical values for the storage capacity of a pit bottom can be based on the shaft-winding capacity. Apart from the accommodation next to the shaft, the storage on the full side must be at least large enough to store a number of tubs corresponding to at least half an

hour's winding capacity, e.g. a shaft with two winding systems, each having an average capacity of 240 tubs per hour, requires a storage capacity of at least 240 tubs. This implies a length of standage equivalent to 5 trains, assuming each train has 50 tubs. With a tub length of about 2 yards, including couplings, this would mean that with two gathering tracks, each track storing 120 tubs, the length of the standage required would be 240 yards. With four gathering tracks, each track taking 60 tubs, the length of the siding would be only 120 yards. Since the width of the roadway required is increased with the addition of parallel tracks and wider roadways are more difficult to maintain, it is generally preferred to restrict the full siding to two tracks, which are split into four tracks near to the shaft to enable decking to be carried out with a double winding system in the shaft.

The empty sidings must be as large as the full sidings, and in addition there must be accommodation for dirt and material tubs. The empty sidings are accordingly larger than the full sidings and are usually from 1·3 to 1·75 times that of the full side, depending on the quantity of shale being handled. Thus, apart from the two sidings for the empty tubs, at least one track is required for stone tubs and another for material tubs. It is sometimes advisable to provide separate tracks for large stones and crushed dirt or washery refuse. The length of the siding should be at least the length or twice the length of a train.

In certain pit-bottom layouts, the storage space is not limited to the full and empty track alone and, if necessary, trains can be lined up in the bypasses. This accommodation has to be considered in designing the pit-bottom layout. Special circumstances have to be considered where a single winding system is drawing coal from several haulage levels. In this case the coal is running continuously to the shaft, but winding from the various levels is intermittent. In such circumstances, the storage capacity at the pit bottom must be large enough to receive the continuous output without interruption while winding is carried out at another level.

(*b*) *Track curves.* The extension of the pit bottom depends also on the radii of the curves adopted. The curves should be larger than those usually used for single-tub traffic, because of the greater friction loss. With single tubs, the minimum curve radius varies from 15 to 35 feet according to the length of the curve; it is usual to adopt

the rule that the minimum radius is seven times the length of the wheel base. In pit bottoms, these minimum radii are not practicable and it is seldom that radii are used of less than from 45 to 60 feet. Where full trains have to be marshalled on a large pit-bottom curve, the radius would not be less than 90 feet.

The probable introduction of large mine cars should be considered when deciding upon the minimum radius for pit-bottom curves.

The planning of a new pit bottom has to start from the shaft. Space for the decking devices and for the track immediately in front of and behind the shaft have to be determined (refer to Section 4), and the length of track added on both sides. The tracks are connected finally by large curves of suitable radius. Where the conditions require it, another curve may be included between the shaft and the sidings for empty tubs.

The roadway junctions when driven in rock should not be acute-angled. Roadways at double-junction points should be staggered. The rock pillar left between two roadways should be adequate in size to have sufficient bearing strength, and this point must be noted when planning the layout of the tracks.

The pit-bottom layout should not extend beyond the margins of the shaft pillar, otherwise additional coal pillars would require to be left for protection.

(*c*) *Width of sidings*. The width of the pit bottom depends in particular on the number of lines of track installed and on their distance apart. The tracks should be laid in such a way as to leave the maximum possible clearance between tubs and between tubs and the roadway sides. In curves, the minimum distance required should be increased and it may be necessary to increase the clearance between tracks where switches are set. The width of the travelling road should not be less than 2 feet and should be at least 6 feet high; it is often advisable to widen travelling roads. In the empty sidings the number of travelling roads should be sufficient to allow good access to the trains on the different sidings.

(*d*) *Height of sidings*. Since ventilation requirements determine to a large extent the size of the roadway, the height of the sidings will depend upon the width. Having determined the quantity of air required in the mine for the output, it is convenient to restrict the air velocity in the pit bottom to a maximum of 1,000 feet per minute. It is advisable to operate with much lower air velocities to avoid the

creation of a dust problem. In the section calculations, the amount of space to be taken up by tubs, pipe-lines, etc., should also be taken into consideration.

It is found advisable to gradually increase the height of the pit bottom near the shaft; in this way the velocity of the ventilating current will decrease in front of the shaft and the air which is usually passing at a fairly high velocity down the shaft can be deflected into the pit bottom. In addition, the increase in height at the shaft is necessary to allow freedom of movement when winding materials such as pipes, rails, etc., and will allow the space to accommodate a landing stage for man-riding, if required, at a later stage.

Section 2. The Handling of Mine Cars at the Shaft Bottom by Gravity

Haulage operations at the shaft bottom and cage decking should be carried out with as few men as possible. Where the number of operators cannot be reduced, work should be simplified so as to allow the employment of personnel unsuited for heavier tasks. Automatic handling of the tubs can be arranged either by gravity methods or mechanically.

Gravity haulage can be used for the movement of whole trains or single tubs and for decking. The system is simple and has the advantage that little or no machinery is required. It has the disadvantage that the tubs suffer more damage because of the lack of continual positive control of movement. Tub collision can result in a great deal of wear, particularly if they are not fitted with spring buffers. The incidence of derailments is high, since colliding full tubs are prone to tilting, due to their high centre of gravity, while empty tubs are easily thrown off the rails when in collision on curves. The degradation of the coal and the production of dust in such collisions are also great disadvantages.

The consequences of such collisions in the case of longer and steeper gradients are very much greater, especially if the tubs are heavy due to their size or full load. An added difficulty in gravity haulage is the maintenance of even gradients over long distances, due to the effect of floor creep.

A simple example of gravity haulage is illustrated in Fig. 347 (a). In this case each track is laid over its whole length at a gradient of 1 per cent. towards the shaft. If there is a single-shaft winding system, it is advisable to have two gathering tracks, one for each cage.

The tubs must be blocked by a locking device of very heavy construction, so that when it is tripped the set of tubs begin moving under gravity. If there are two tracks for a double-winding system,

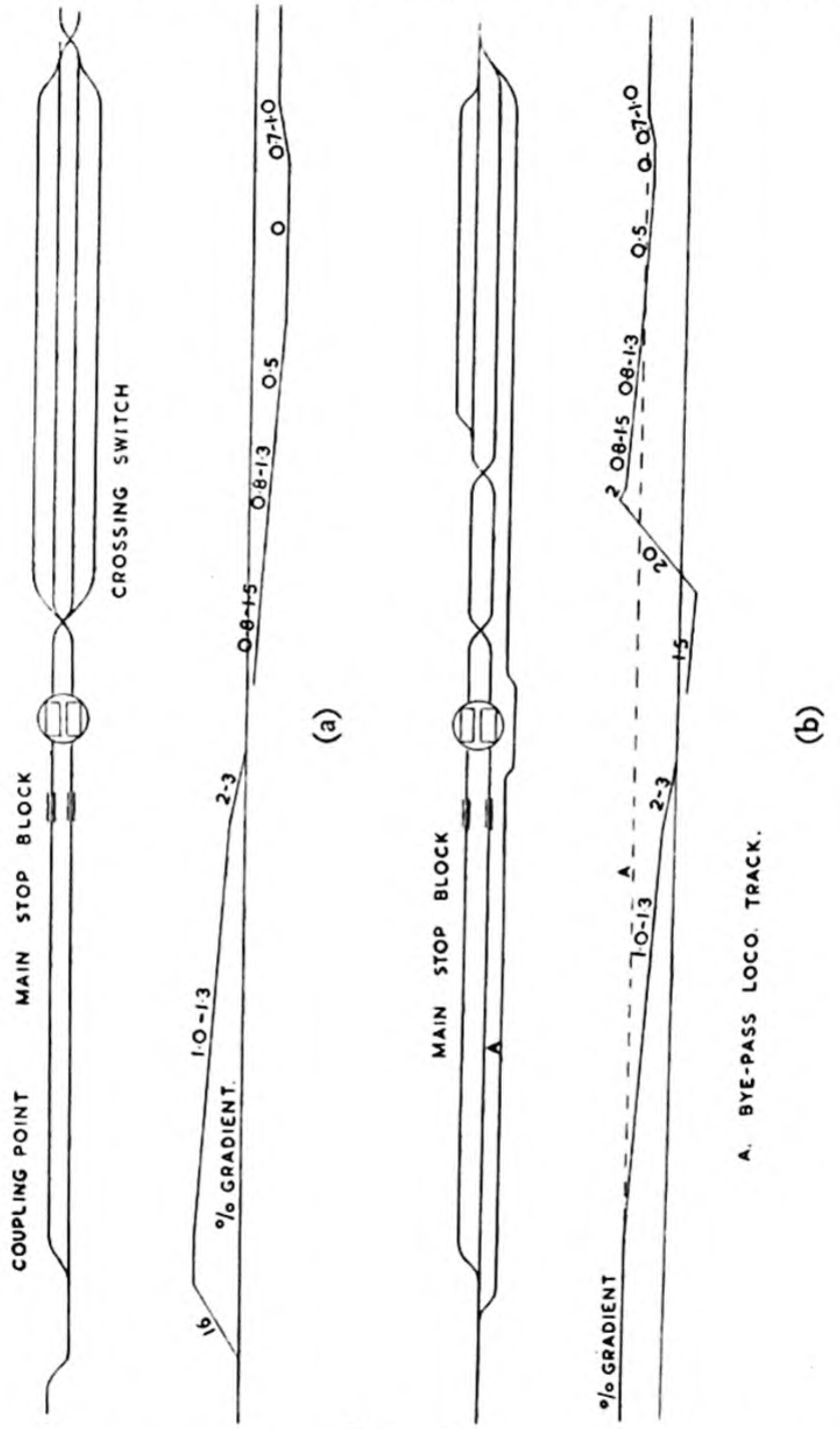


FIG. 347

in which four cages are used, the individual tracks must be similarly equipped and strong enough to ensure holding the complete train of tubs.

The reduction in roadway height associated with this system of

haulage can be overcome by guiding the track over a kip or shunting incline, as shown in Fig. 347 (b). The full train is pushed up an incline at the end of the level siding, using either the main-road locomotive or a special shunting locomotive. At the top of the incline the train is uncoupled and the tubs run under gravity to the decking track, which should be fairly long. The disadvantage of this method is the delay to the main-road locomotive if it is used for shunting. The cages are decked intermittently, the tubs being marshalled in the feeding track where they can be stopped with the main stop-block, which is constructed to release only the number required per deck. The track length in front of the stop-block is shown in Fig. 347 to have a gradient of from 2 to 3 per cent. and should be at least two tub-lengths long. The tubs are sent away from this length at a fairly high speed, running straight into the cage and pushing out the empties. There is no possibility of regulating the speed of the tubs to the loading platforms at the shaft, and the decks have to be loaded by hand. In this position the platform is steep and the velocity of the tubs is much too high when decking the last platform, since the rope is now stretched and the platform is inclined downwards.

The empty tubs in the cage are ejected by the full ones and run over swinging platforms into the discharging tracks fitted with high guide rails, the gradient being from 1 to 1·5 per cent.

The centre of movement of the swinging platforms on the empty side is from 4 to 8 inches deeper than those on the full side, according to the depth of the shaft and the load carried. As a result, the position of the platforms is still level or slightly inclined upwards, even when the cage lowers, due to the stretching of the rope. The tubs therefore run off easily. Return locks are provided to prevent the tubs running back into the shaft.

Arrangements must be made behind the shaft to distribute the tubs into the various tracks in the empty sidings. With gravity haulage, the tubs run into the siding at a gradient of from 0·8 to 1·3 per cent., the gradient gradually decreasing thereafter. In the last part of the siding the empty track is level, after passing over a gradient of from 0·7 to 1 per cent. against the load, to prevent the tubs from rolling through. It is not advisable to install brakes at this point because the locomotive would be unable to pass.

On the empty side, gravity haulage presents serious difficulties, especially if the tubs reach a fairly high speed. This disadvantage is

pronounced when shunting heavy dirt tubs. It is necessary also to couple the tubs up individually in the standing train.

Considering these difficulties associated with gravity haulage in pit-bottom layouts, and the fact that simple and reliable mechanical devices are available, it is understandable that the trend is towards layouts incorporating complete mechanical handling of the tub circulation except perhaps immediately adjacent to the shaft, where short inclinations are still used.

Section 3. Mechanical Equipment for Handling Mine Cars

Horizontal chain-creepers are commonly used in pit bottoms, and a design of German manufacture is shown in Fig. 348. The creeper

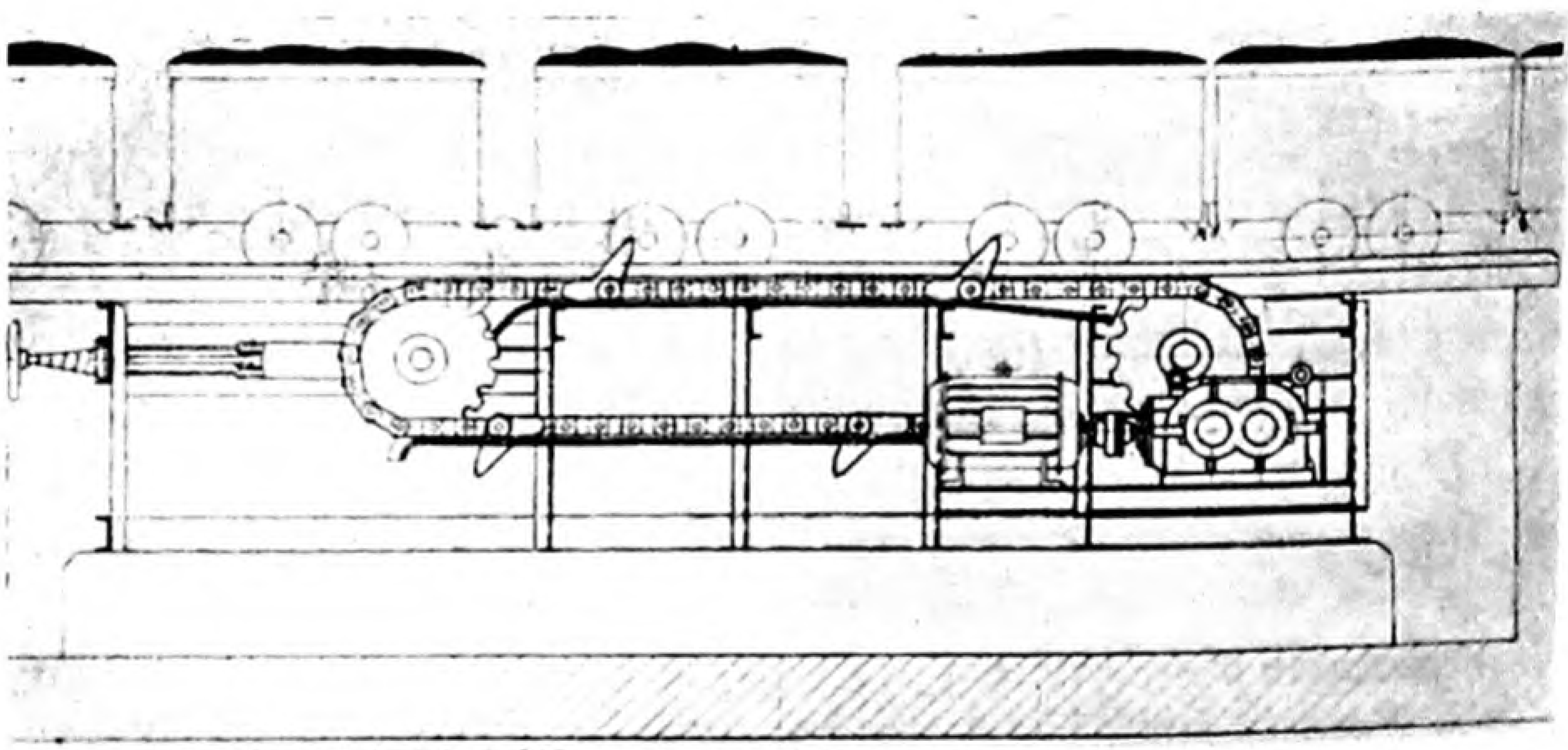


FIG. 348

consists of an endless sprocket chain, sliding between guides at a speed of about 60 feet per minute over several chain wheels, one of which is the driving wheel and another, fixed in a guide, acts as a tension device and shock absorber. The chain is fitted with single or double catches according to the pulling power. The catches hold the tub by the axle-box cases or by special cams fitted beneath the tubs, the latter arrangement being used exclusively on larger mine cars. Chains fitted with single catches must run out of the track centre to avoid catching tub couplings. The space between the catches is something less than the length of a mine car, including the couplings, in order to slacken the tub couplings when the tubs are running on to the chain-creeper.

If the catches are connected rigidly to the chains, then travelling speed will increase at the reversing drum and a pulsating load will be

transmitted to the tubs, causing a high stress in the axle boxes, in particular at the tail or return wheel. In practice, additional chain wheels are used and rigid catches avoided. The revolving catches fitted are designed so as to run through a guide channel along the chain and to pass a special compensation curve at the end of the creeper. The speed is reduced by the catches tilting back, whereas on the drive wheel the catches accelerate. This device makes the creeper run quite smoothly without shock. The creeper shown in Fig. 349 is the Hausherr chain-creeper, in which the catches turn down completely before reaching the return wheel. All horizontal chain-creepers are fitted with guide rails, while top guard plates cannot be used if the locomotive is to run over the same track.

Electric drive is preferable, the creeper being driven through a bevel or worm gear and a flexible coupling. Generally there is another spur-reduction gear or a cone belt between the first gear and the coupling. Pneumatically operated claw couplings or coil-and-disc clutches are designed for stopping the creeper while the motor is running. It is also possible but expensive to install planetary gears which begin to slide in the event of an overload.

(*a*) *Caterpillar chain-creepers.* Instead of the horizontal chain-creepers, the 'mine-car transporter' may be used. The general design of this type, as shown in Fig. 350, consists of two caterpillar tracks running in a robust framework. The wheels of the mine cars rest on the caterpillar tracks in the same way as on rails, the tub wheels being pressed down on the caterpillar track by a spring-operated bar. The speed of the caterpillars must be twice the travelling speed of the mine car.

With this form of transporter, differences in the wheel diameter can be accommodated. The pressure bars can be lifted by jacks, if necessary, to release the mine cars. The transporter may consist of one or several units, the pull of one unit being at least 1,300 lb., and when travelling at a speed of 60 feet per minute, it requires from 6 to 7 h.p. On level roads a three-unit transporter can adequately deal with 100 average-size mine cars.

An advantage of this form of transporter is that it operates without any shock to the travelling mine cars, and maintenance of the cars is consequently reduced. The transporter can also act as a brake, through sliding friction, when some obstacle in front of the transporter stops forward movement of the mine cars.

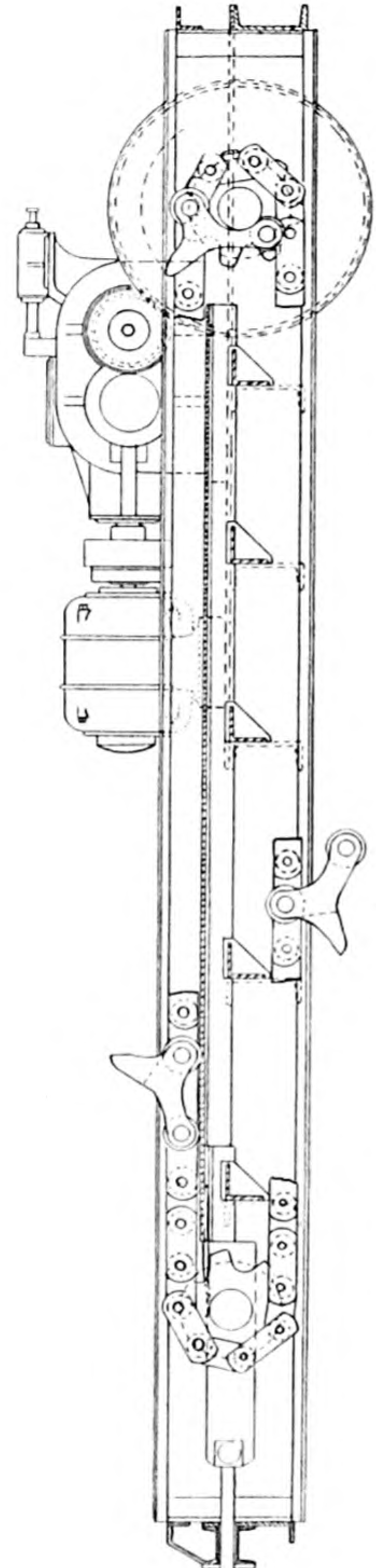


Fig. 349

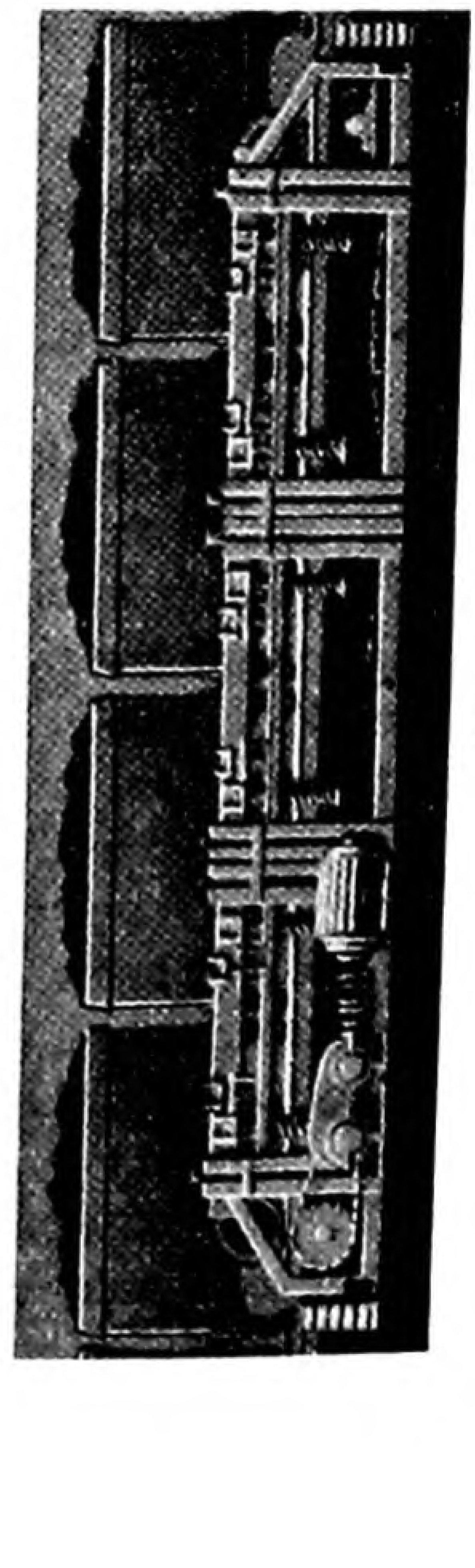


Fig. 350

(*b*) *Compressed-air-operated rams.* It is possible to push or pull forward mine cars using mechanically operated rams or pushes. Continuous rams can only be used in pit bottoms. An inherent disadvantage is that they must be operated by compressed air, and electrical power is not possible, and, in addition, shocks to the cars cannot be avoided. A compressed-air-operated ram is illustrated in Fig. 351. The tractive effort of these rams ranges from 3,000 to 6,000 lb., according to the cylinder diameter.

(*c*) *Retarders.* On the full sidings it is the purpose of mine-car brakes or retarders to stop complete trains or single tubs in con-

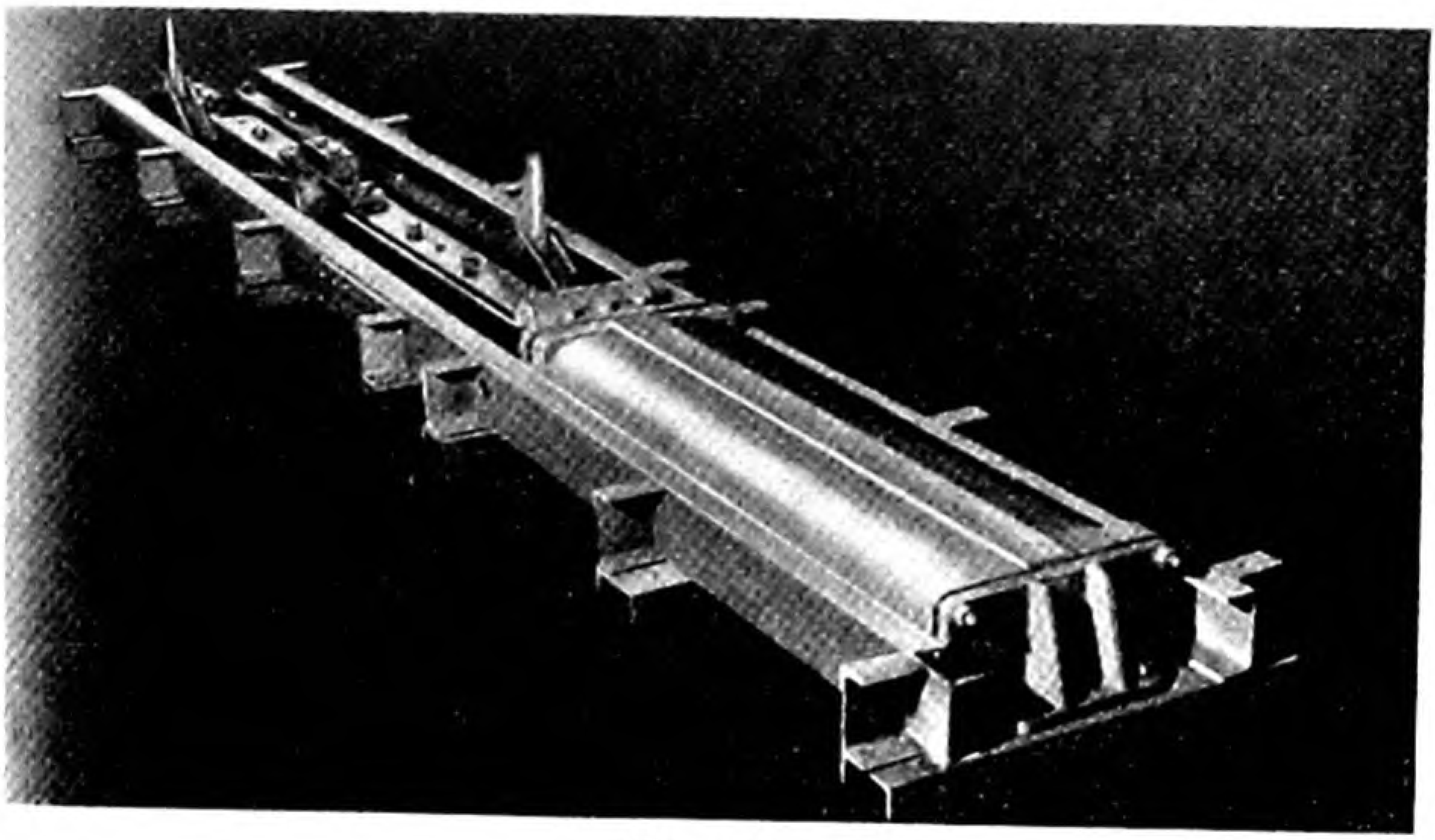


FIG. 351

junction with tub locks as far as is practicable without shock. On the empty sidings it is their task to stop or retard dirt tubs, which generally pass the shaft at a high speed. In order to brake the cars without shock, the braking distance should be as long as possible.

The brakes are usually operated by compressed air, weights or springs, and are generally released by compressed air. The speed of smaller tubs is usually reduced by applying the brake blocks over the wheel-treads and, consequently, rapid wear of the treads takes place. With large mine cars the brake blocks are applied to the inner side of the wheel or to the body of the car, which is fitted with a special braking surface. The latter system of braking is preferable, since it does not affect the wheels. A German design is shown in Fig. 352.

Attention should be directed to the special form of braking applied to fast-moving cars, e.g. dirt tubs, with the object of reducing excessive speed, while giving passage to cars running at low speeds. One form of this type of brake is based on the action of the fast-moving axles of the tub against the catches of a chain-creeper. The chain-creeper is started by the car and in turn drives an oil

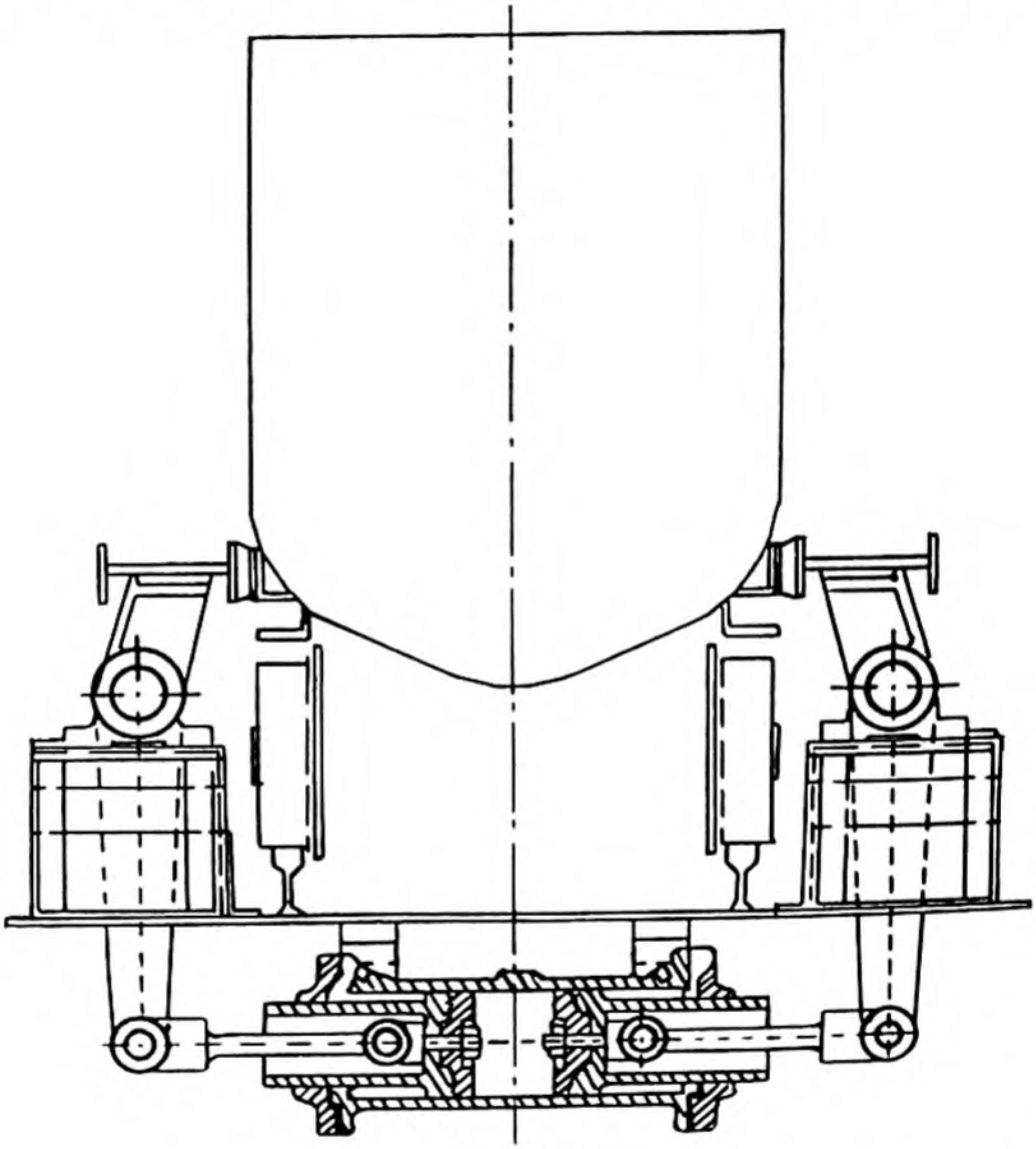


FIG. 352

pump which operates an inside shoe brake on the chain drive, thus reducing its speed and consequently the speed of the mine car.

(*d*) *Locks or buffers*. The use of the term 'mine-car lock' implies the application of catches to the front axle or the buffer of the mine car and thus blocking its movement. A flexible design incorporating springs or an air cushion is usual with such locks in order to prevent the tubs or the locks from being damaged by sudden shocks. It is possible also to reduce the speed of the car before reaching the lock by using a car brake or retarder.

There are two kinds of lock, namely main locks and shaft locks. The purpose of main locks is to stop the tubs in front of the shaft and to separate the number of tubs necessary for one deck of the cage, so that when the lock is tripped, the tubs run by gravity towards the decking device. The use of brakes without locks is not recommended at this point because of the danger of tubs running through to the shaft.

The main locks can be of the single or double type, the single-cam locks being tripped manually and designed to close automatically when two tubs have passed. The single-cam locks are shorter than the double locks which are fitted with a front and tail cam working in opposite directions. When the front cam is lowered, one or two tubs, according to the capacity of the cage deck, are released and run by gravity towards the cage. The tail cam is lifted simultaneously and retains the following tubs. The distance between two cams is equal to the length of one or two tubs, whichever number is to be separated.

Usually the front cam retains the tubs by the buffer, whereas the tail cam can hold them only at the axle or at special cams below the tubs, this latter system being common with large mine cars.

Since flexibility of the lock is very important, either the whole lock or each cam is separately cushioned, allowing a movement up to 2 feet.

A type of mine-car lock is illustrated in Fig. 353, in which a beam with a cam at both ends is borne on a sliding block running on rollers, the slide block in turn being coupled to an air-cushioned piston rod. The lock is operated by the tilting of the beam, which is top-heavy. This movement is started by the supported rollers running on rising cylinders and these are pushed forward under the top of the beam or retracted by a compressed-air piston. Should the compressed-air supply be stopped, the rollers do not run back automatically. The lock is fitted with guide plates over the wheels in order to prevent the tubs from tilting or derailment. The device can take great strains without difficulty.

The shaft locks are necessary in association with rams, in order to stop the mine cars running towards the cage at the point where they are to be picked up and pushed forward by the ram. The shaft locks should be as near as possible to the shaft so as to reduce the space over which the ram has to move the mine cars. The braking

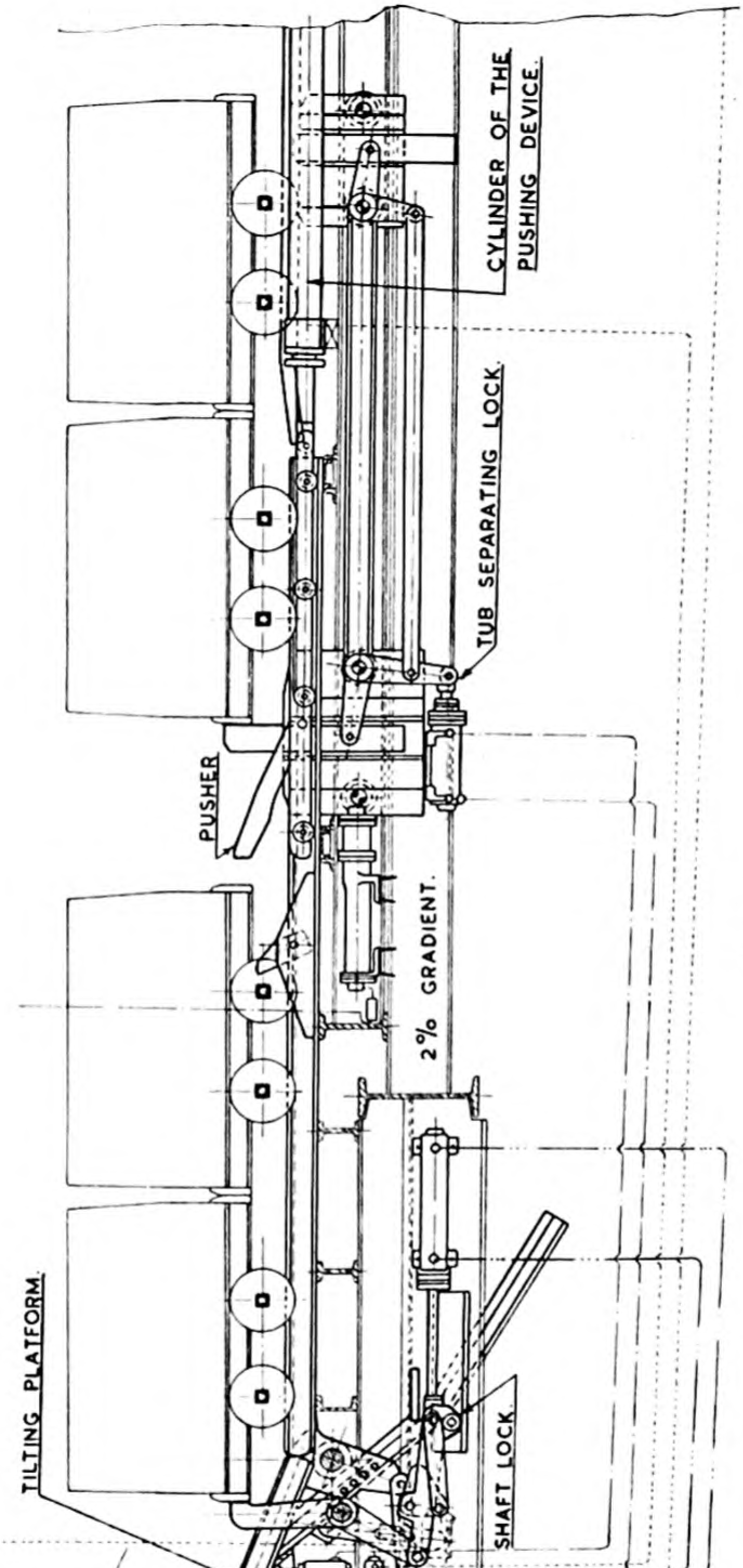

CYLINDER OF THE PUSHING DEVICE.
TUB SEPARATING LOCK.
PUSHER
2% GRADIENT.
TILTING PLATFORM.
SHAFT LOCK.

Fig. 353

effect which the shaft locks have to produce is rather limited; they may hold back the tubs by the axles, wheels or buffers, though the last point is preferred. In the case of a failure of the compressed-air supply, a special safety device should be incorporated. If the lock is cushioned through an air piston, this cylinder may be used also for operating the lock. In some types the brakes operate on the top of the wheel-treads.

A shaft lock which is incorporated in a tilting platform is shown in Fig. 353; in this case the distance between the lock and the cage is very short.

(*e*) *Automatic decking equipment.* The essential feature of decking devices is an electrically or compressed-air-operated ram which

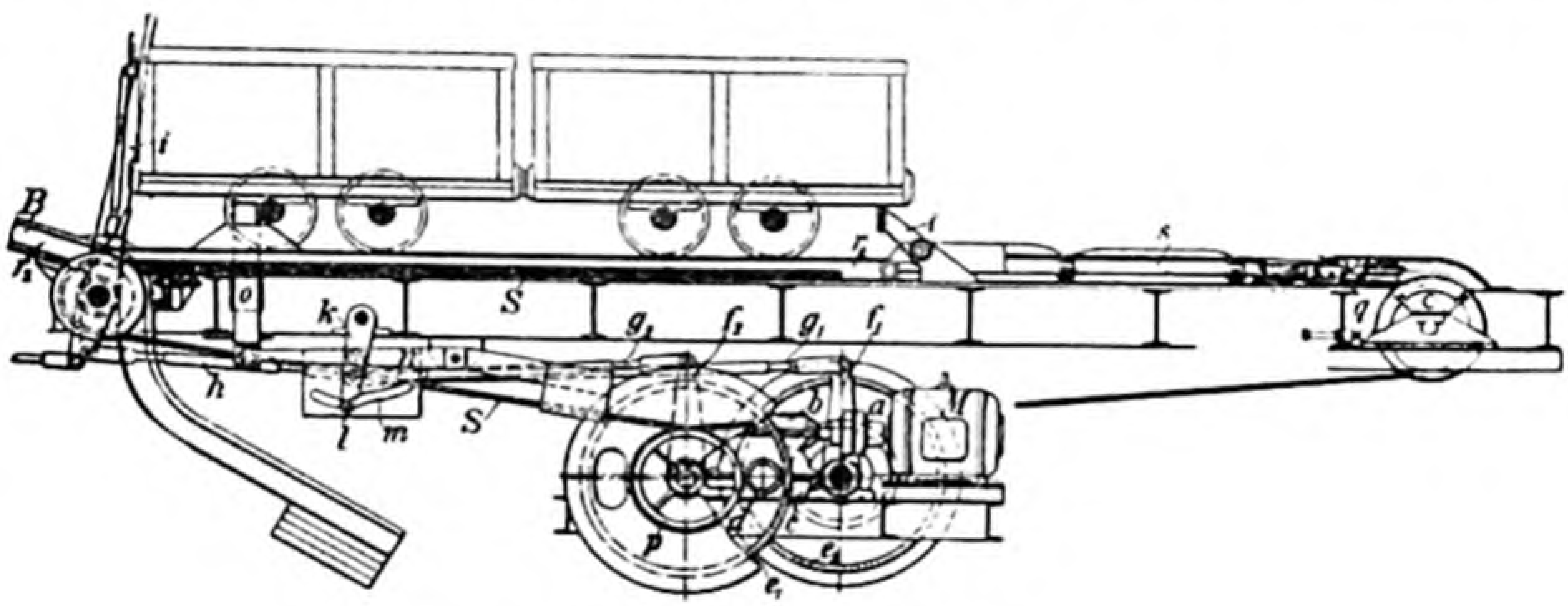


FIG. 354

attacks the rear of the tub. The ram is usually fastened to a sprocket chain and runs in a guide-way between the rails, the chain being sufficiently flexible to follow the movement of the tilting platform.

It should be possible to regulate the chain speed in such a way that the impact on the rear of the tub is not too great, and so that it develops a greater pushing force as the tub ram moves forward. The mine cars on the cage deck should not be pushed off by a sudden shock but by a continuous movement. On the return stroke, the pusher must tilt down in order to pass under the tubs without touching the axles. A well-designed, electrically operated pusher is shown in Fig. 354. The continuously running, squirrel-cage motor drives a small friction wheel through a reduction gear. The friction wheel is coupled alternately with two large pulleys, causing the rope drum to turn in opposite directions. A rope running over guide

pulleys connects the drum with an articulated rod or a sprocket chain bearing the pusher.

Compressed-air-operated decking devices consist of a long cylinder installed between the rails. The top of the heavy piston rod is connected by a sprocket chain with the pusher. The initial cost of electrically driven devices is higher, but the operating cost is less and they have not the operating difficulties associated with compressed air. The electrically operated type are preferred at the surface, but they require more space.

(*f*) *Tilting platforms*. Tilting platforms are used to compensate for the difference in level between the pit-bottom rail level and the cage deck caused by the rope stretch on loading the cage. The tilting deck allows the cage to be loaded and unloaded while the winding rope is still oscillating vertically without losing time. When Koepe winding is adopted this form of decking is essential, since keps cannot be used or rope slip would occur, whereas with normal drum winding kep gear is usually installed.

The difference in level to be accommodated by the platform may be 6 inches or even more; the greater the difference, the longer is the tilting platform required. In the latter case the decking time is increased and it may be advantageous to compensate for the difference in rail level both at the surface and the pit bottom by installing tilting platforms of half the required length at both points. Where large mine cars are being used, the longer tilting platform is advisable.

The maximum difference of level which can be accommodated by the platform must be calculated, taking into account the lowest level on the loading side and the highest level on the unloading side of the shaft decking platforms. The lowest practical position of the loading platform which is possible depends on the maximum inclination and the fact that the platform tongues must overlap and rest on the cage deck by at least one and a half inches. The highest practical position on the empty side is governed by the fact that the platform tongues must find the necessary support on the cage deck. There must be sufficient space between the cage and the tilting platform while the latter is in its highest position. When calculating the maximum difference in level which can be compensated in practice by the platform, it must be remembered that its centres of movement are not at the same level.

The diagram in Fig. 355 shows that the tilting platform consists

of swinging rails covered with steel plates. Tongues fastened to the front of the platform rails can be tilted down by the descending cage, while the ascending cage can lift the platforms until it can be accommodated in contact with the deck. Counterweights may be used, but they are often disliked because they create difficulty by obstructing the man-riding level immediately below. Generally the tilting platforms are operated by compressed air, each platform being separately

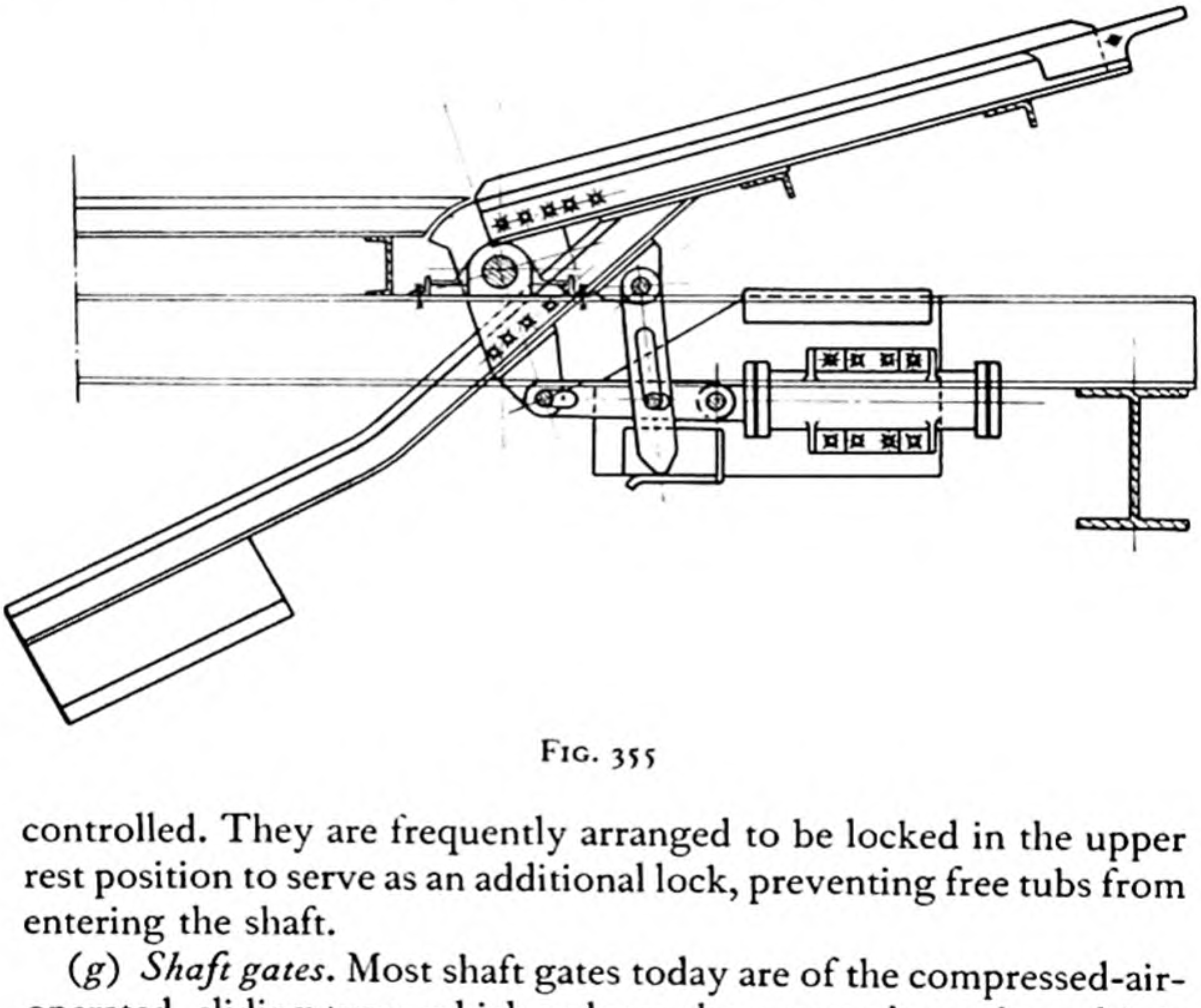


FIG. 355

controlled. They are frequently arranged to be locked in the upper rest position to serve as an additional lock, preventing free tubs from entering the shaft.

(*g*) *Shaft gates*. Most shaft gates today are of the compressed-air-operated, sliding type, which reduces the onsetter's work, and it is no longer necessary for him to be stationed immediately at the shaft side. The gates on the full and empty sides can be opened and closed simultaneously and the system allows the doors to be interlocked while the cage is travelling in the shaft.

The shaft gates may be moved either directly on a level rail by a compressed-air-operated ram, or they may run down by gravity on an inclined slide rail which is lifted by a compressed-air device. The design shown in Fig. 356 has the advantage that air-cushioning protects the doors from shock when reaching their final position.

(*h*) *The operation of automatic decking equipment.* During shaft winding the following devices have to be operated in sequence: shaft gates, tilting platforms, shaft locks, loading rams, tub-separating lock. Generally, the same lever operates the tilting platform, the shaft lock, the rams and the cage gates, but there should be a time-lag between each operation, the swinging platforms moving first, the shaft lock next and the ram last. When the cage has been decked,

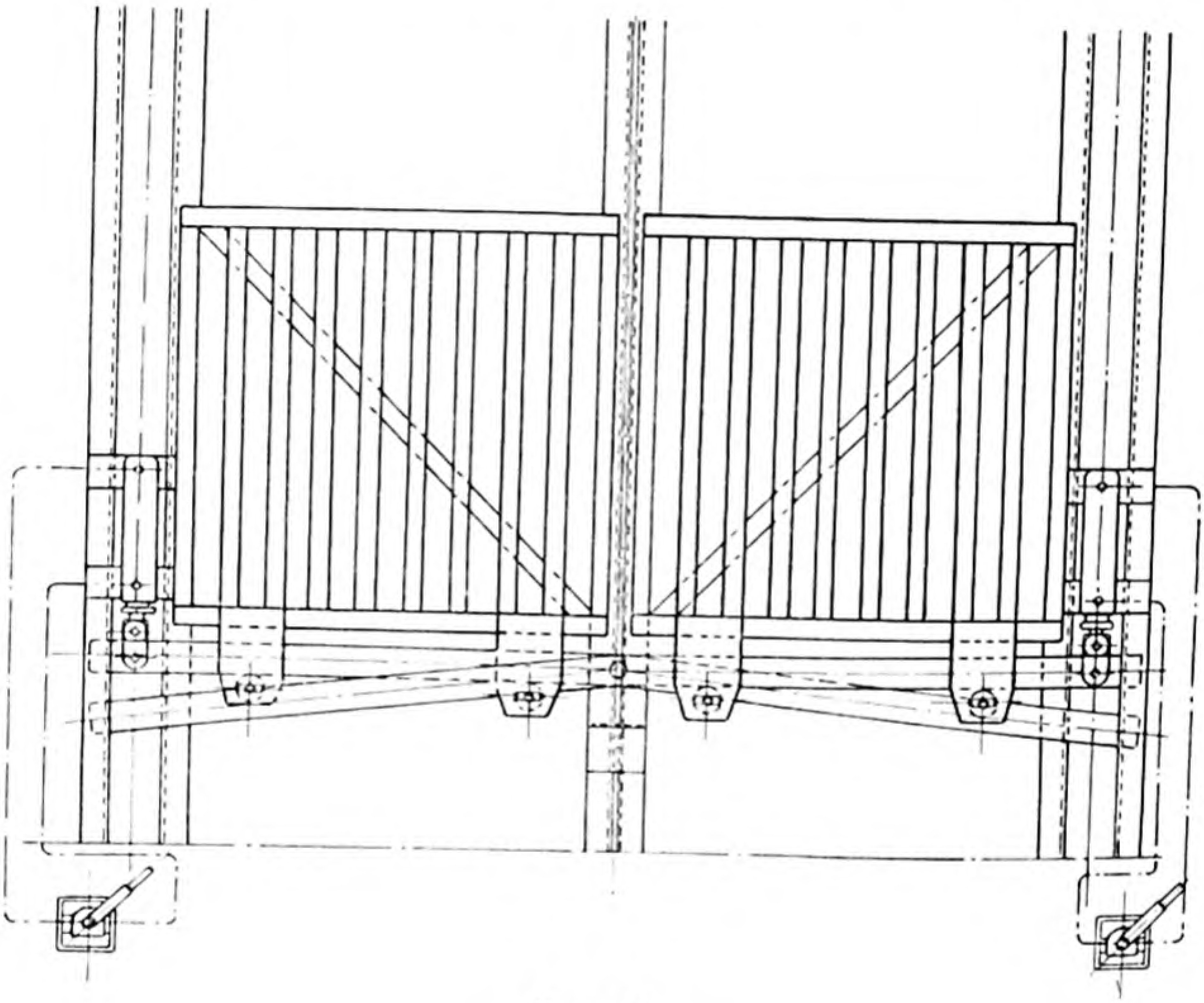


FIG. 356

the shaft lock should be closed again as quickly as possible. During the time the cage is changing decks, the shaft gates, which are operated by a special lever, remain open. The lock which separates the tubs to be decked is generally operated independently from the other devices.

During man-riding only the shaft gates and the tilting platform need to be operated, and as the gates have a separate valve control, the operation of the tilting platform must be separated from the ram and the main lock. It is sometimes preferred to have the shaft lock

individually operated so that single tubs, such as for materials, can be pushed by hand through the cage.

The different devices are operated by stop-valves installed within easy reach of the operator and they can also be released by another control lever which operates the individual valves connecting the various devices. When this lever is moved towards the shaft, successively the swinging platform, shaft lock and ram come into operation.

All the decking equipment rests on a support structure which is illustrated in Fig. 357. Three cross-beams support horizontal joists which are covered with steel plates. The advantage of the girder

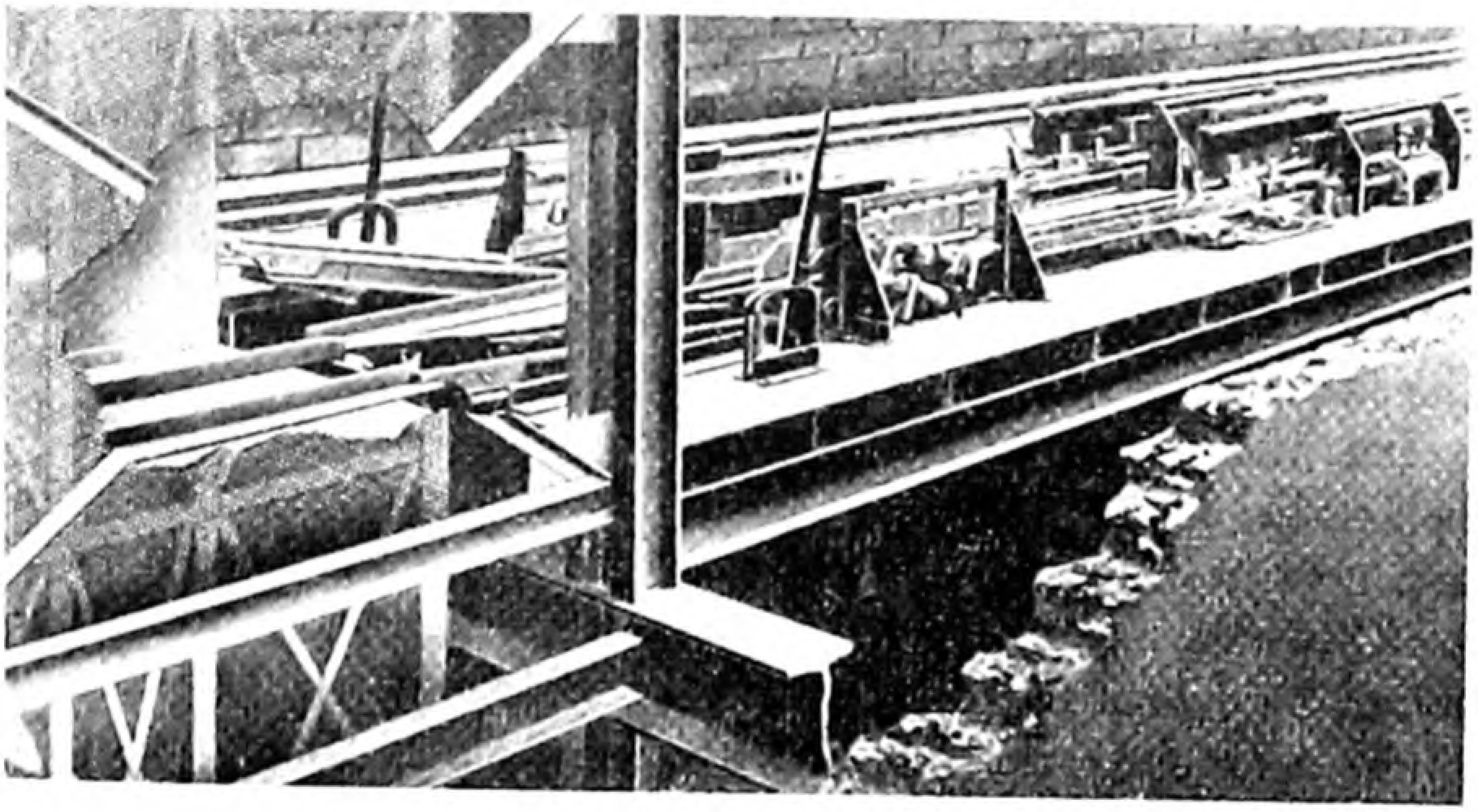


FIG. 357

framework lies in the fact that the equipment is not so easily affected by strata pressure and movement. It is possible with this contruction to assemble the framework and decking equipment at the surface, making it easy to install it quickly underground.

Section 4. Principles Involved in Planning Shaft-bottom Layout

Pit-bottom planning is based generally on the principle of one-way traffic, the cars being pushed towards the shaft, in which case the pit bottom is divided into two sections. On the full side the cars are pushed into the cages, while on the empty side the empties run from the cage.

Accordingly, when planning a pit-bottom layout, the arrangements must include provision to marshal all the full trains from

different directions at the full side. On the other hand, it must be possible for the empty trains to start from the empty side to all districts of the mine. Finally, it must be possible for the locomotive to get to the empty side after being uncoupled from the full train in order to take over the empty return trip.

(*a*) *Layout of full and empty sides.* The *full side* should always be near to the shaft. It is an advantage from the point of view of supervision to have the track laid in the direction of the shaft axis. However, this is not necessary, since long trains can be pulled round a curve by a chain-creeper. With a curve of large-enough radius, a change in direction of up to 180 degrees is possible. It is for this reason that the extension of the full side of a pit bottom is often reduced, the coal trains being stopped at the curve, which is required in any case. Should the output be drawn from both sides of the shaft, each side has a similar siding and in this case the dimensions of the layout must be kept as small as possible. The two full sidings are connected just in front of the chain-creepers taking the full tubs to the shaft.

It is useful practice to have the *empty side* as near as possible behind the shaft. If these sidings are located too far from the shaft and gravity haulage is used, the tubs would need a long gradient, which would require to be fairly high if the straight section is followed by a long curve between the shaft and the empty sidings. Quite apart from this consideration, an inclined track cannot be used for gathering together trains and single tubs running by gravity through a curve, as they are difficult to supervise and are a source of danger and possible accident.

Creepers on the empty side must be of the pushing type, since the tubs cannot be recoupled before being distributed into the several tracks. In this case a slight gradient is sufficient in the curves, but it still reduces the overall clearance of the track. Another disadvantage of the inclined curve is the difficulty in reaching from the empty side a long line of tubs standing in the curve. Thus, it is obvious that a long distance between the shaft and the empty sidings has great disadvantages.

The empty siding should not be curved, so eliminating the possibility of empty tubs running under gravity and knocking one another and being derailed. The siding should be close to the back of the shaft and in line with the shaft axis. If this principle is observed,

it will not be possible in every case to have an empty siding for both sides of the shaft if coal is arriving from both sides. A disadvantage of separate empty sidings on each side, however, is that the empty cars are split up and it is inconvenient in the event of a shortage of empty tubs. It may be possible that there is a full train of tubs available, distributed on both sides and it is impossible to make up a train. This disadvantage may be overcome by systematic distribution of the empty tubs.

(*b*) *Traffic organisation*. When planning a pit bottom, the whole system should be based on the principle of one-way traffic, implying a special track for both directions. The running of trains in opposite directions on different tracks in the same roadway should be avoided at peak periods and if possible at all times.

Traffic circulation in opposite directions occurs if there are no loops and the whole pit-bottom traffic is in one roadway. In the narrower sense, trains passing the shaft will always run in the opposite direction to the pit-bottom traffic, and in this way it is difficult to supervise the traffic and the danger of accidents is increased. The shaft personnel is disturbed by the passing trains, additional track is required and the shaft bottom has to be wider to accommodate it. This layout is particularly troublesome at the shaft and in the empty siding, where several tracks other than this shaft bypass are required. It should be noted also that varying gradients are necessary within the pit bottom affecting both the traffic and the supports and may also aggravate the incidence of derailments. The need for these gradients is discussed in a later section.

Tracks should not be crossed if at all possible, since in this case trains will have to reduce speed, stop and start again, thus slowing down the continuity of the haulage. A junction of two tracks is not serious if these tracks are running parallel over a certain distance. Where crossings and junctions are unavoidable, the difficulty must be overcome by simple and clear regulations referring to the right of way or by signals.

Where a single control point for the switches at the entrance to the pit bottom is incorporated in the layout, the signals and water sprays should also be controlled from this switch cabin. In any case the traffic control should be as simple as possible and easy to supervise.

The system adopted for the arrival and departure of trains is the most essential feature of the pit-bottom layout. These systems vary

according to whether the continuation of the pit bottom is straight, curved or has a reversing track in which the whole train stops in order to return in the same direction. In the latter case, the locomotive returns pushing the train. If the full trains enter the pit bottom from a straight roadway or a curve, they are generally drawn into the full siding by the locomotive. This can be done in two ways as shown in Fig. 358 (a). In the first case the locomotive may be left in front of the train to pull it over a horizontal chain-creeper or pusher. After the locomotive has been shunted over a switch in front of the creeper into a siding in order to return to the loading-station, the full train is moved forward by the first chain-creeper to a second creeper.

In the second case shown in Fig. 358 (b), the locomotive is shunted into a siding when reaching the pit bottom; from this point it draws the train forward by means of a chain to a second waiting train. The full tubs are then moved forward by a chain-creeper. This system has the advantage that only one chain-creeper is required. The system illustrated in Fig. 358 (c), in which the locomotive pushes the full train into the siding, is not often adopted. In this case the locomotive is uncoupled before reaching the pit bottom; returns to the rear of the train and pushes it up to a chain-creeper which takes it forward. The adoption of this system depends on the particular conditions but has the advantage that no special locomotive track is required on the full side.

Should the trains come from a reversing track as illustrated in Fig. 358 (d), the locomotive usually pushes the train into the pit bottom without changing its place. This system has the advantage that no special locomotive track is required and the locomotive does not change its position on the set. On the other hand, reversing tracks usually require a longer pit-bottom layout.

The only fully satisfactory system of fetching *empty* trains from the pit bottom is for the locomotive to be placed in front of the train and to pull it forward. Reversing tracks on the empty side are not at all suitable, while their use on the full side dispenses with a special locomotive track as shown in Fig. 358 (e).

In order to obtain the full capacity of the locomotives, the trains should leave the shaft bottom as soon as possible, so that they deliver the full train rapidly, run quickly from the full to the empty siding and take over the empty train for the return journey without delay.

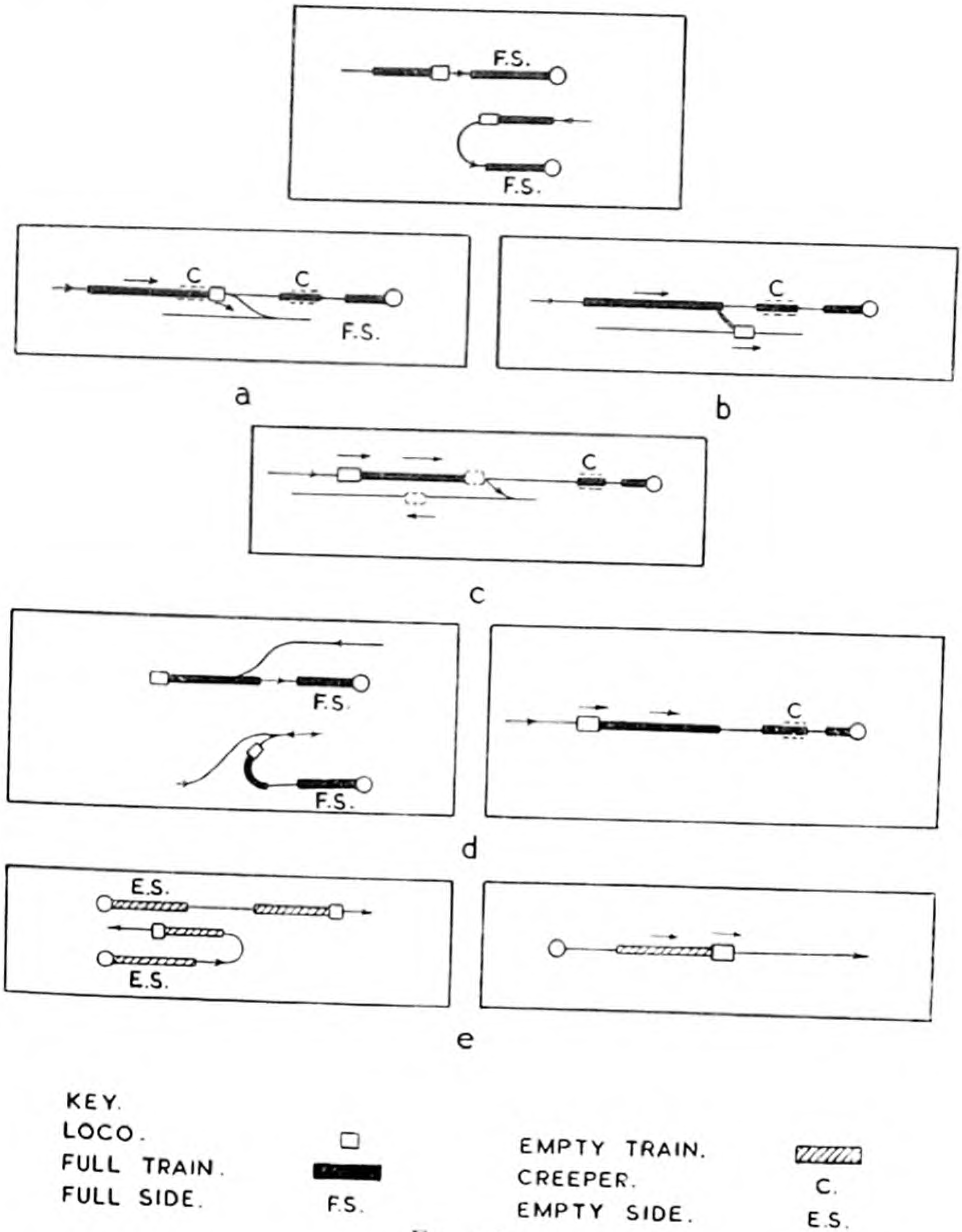


FIG. 358

The locomotive may leave the full train earlier if the shunting operations are carried out by a special shunting locomotive. It is better to do without such a locomotive if possible and to dispatch the arriving trains as quickly as possible with the service locomotive.

When leaving the full train at the entrance to the pit bottom, the locomotive is located either at the end of the train on the gathering track or on a special locomotive track beside the train. Where two

gathering tracks are in use, one locomotive track in the centre is sufficient.

The distance the locomotive has to travel from the full to the empty side should be as short as possible in order to save time; crossings, a great number of switches and the breaking of the one-way traffic principle during an idle locomotive journey are less serious than in the case of complete trains, although these factors remain a source of trouble to a certain degree. These difficulties are much more serious if the track is not easy to supervise and the trains follow one another very quickly.

There are several methods for running the locomotive to the empty side. The distance it has to cover is short when it passes the shaft, in particular if it takes the full train to the shaft bottom. A special locomotive track, however, results in wider sidings and shaft landing, and to avoid this necessity a small curve or pass-bye for the locomotive may be provided.

If the locomotive passes the shaft on the same track used by the full or empty trains, running in the opposite direction to the main traffic cannot be avoided.

Unless the locomotive loop track is introduced, the return trip to the empty side is long and can be made shorter only if the full trains are pushed into the pit bottom by the locomotive, and the entrance to the full siding and outlet to the empty siding are not too far apart. In the loop, the direction of the locomotives is opposite, in any case, to the direction of the trains and, since these loops are difficult to supervise, a special locomotive track should be laid.

The third method which it is possible to provide is a special loop exclusively for the locomotives and in this way it travels the shortest distance from the full sidings to the starting-point of the empty sidings.

The empty tubs must be coupled together when waiting for the locomotive, so that on its arrival the train can start without delay.

(c) *Examples of pit-bottom layout.* The form of a pit bottom depends on the layout of the track, which in turn is governed by the principle that each haulage track must be connected with both the full and the empty side. To meet this necessity, the track system must include a loop or a reversing track.

Different forms of the pit bottom are associated with each track

layout according to whether the shaft is situated in the axis of the main haulage road or outside, and whether the full trains reach the shaft from one or from both sides. The output also plays a part in deciding the layout. Generally the shaft is in line with the main haulage road. If all the full trains arrive from the same side, it is possible to use the enlarged roadway as a pit bottom and to equip it with a reversing track or to have a loop-shaped pit bottom. The roadway pit bottom with a reversing track is particularly suitable with a single shaft-winding system dealing with a small output, and it is cheap. Three different forms are illustrated in Fig. 359 (a). With the systems a.1 and a.3 the trains and the locomotives have to pass by the shaft, the latter running in the opposite direction. This disadvantage may be obviated by having a separate locomotive track; in this case, however, the width of the pit bottom is increased. The advantage of system a.2 is the very short extension of the pit bottom and the short journeys for the locomotive.

In the case of larger outputs, the loop pit bottom is preferred and is very common if all the coal trains are coming from one side. Three different types are known, as shown in Fig. 359 (b), system b.3 being the best. In this case, the curve is used as a full siding, which results in a short extension of the pit bottom. The empty siding is immediately behind the shaft. The locomotives which have drawn the full trains must go back through the curve in order to get to the empty siding or they must pass by the shaft. With system b.2, the length of the pit bottom is about the same; the advantage of this type is the short travel of the locomotives from the full to the empty side. On the other hand, this type has the disadvantage that the siding for the empties is very far from the shaft. System b.1 is the most expensive of all, due to the length of the pit bottom, but it has the advantage that the station for the empties is near the shaft.

If the shaft axis is more or less at right angles to the main haulage road, only the systems shown in Fig. 359 (c) are possible. Of these types, c.2 is more expensive than c.1; on the other hand, the latter has the advantage that the empty side is immediately behind the shaft. The locomotives return the longer way through the curve or they reach the empty side by going past the shaft.

If the full trains arrive from both sides of the shaft, either a pit bottom with two reversing tracks or with double loop tracks must be considered.

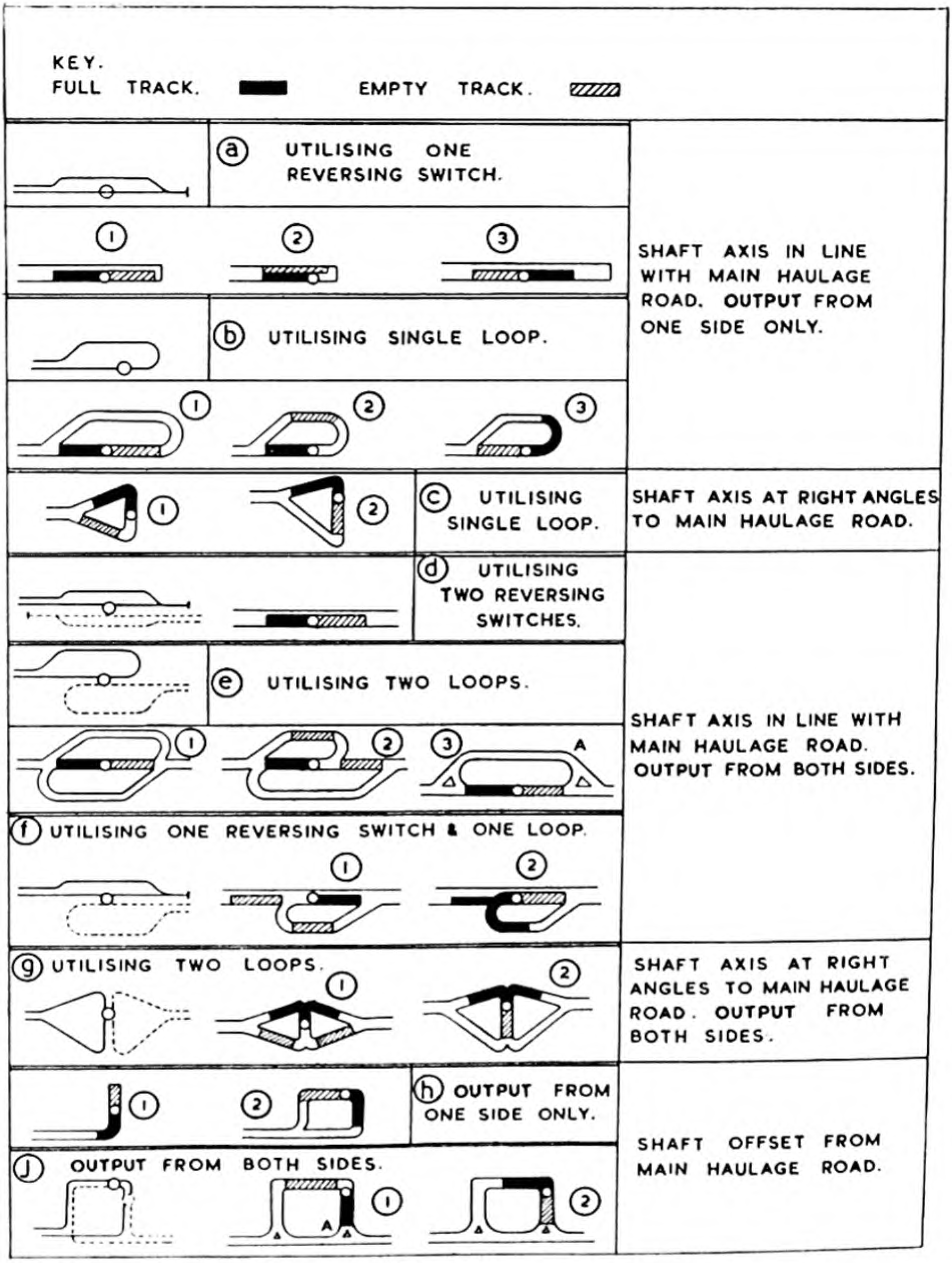


FIG. 359

The advantage of the pit bottom with two reversing tracks is the reduction in cost, but there are other disadvantages. A high output cannot be dealt with and it involves a great width of pit bottom, because full and empty trains as well as the locomotives must pass

by the shaft. As the direction of the locomotives is opposite to that of the train, it is recommended to have an additional locomotive track. The layout is shown in Fig. 359 (d).

For shafts with a high output and with the trains arriving from both sides, the pit-bottom layout shown in Fig. 359 (e) is suitable. The type shown in e.1 is very expensive, the type in e.3 is cheaper and in most cases it is preferred because it is equally efficient. The direction of the full and the empty trains passing through the loop is the same, and it is unavoidable that full and empty trains must run on the same track at point A; a special loop track is necessary for the locomotives.

The advantage of the double-loop pit bottom shown in Fig. 359, e.1 is that both the full and empty trains may run in either direction without interfering with each other. In the case of system e.2 this is most marked, because no crossings are necessary. On the other hand, type e.2 has the disadvantage of requiring two empty sidings, one of them being far from the shaft.

As can be seen by Fig. 359 (f), pit bottoms including both a reversing track and a loop track are possible. Type f.1 of this system has the disadvantage that one of the sidings for the empties is far from the shaft. Type f.2 is most suitable if the output is very high from one side of the shaft and rather limited from the other.

If the shaft axis is at right angles to the main haulage direction, only the pit-bottom layout shown in Fig. 359 (g) is possible. Type g.2 is more expensive than type g.1, but the latter has the advantage of having only one empty side adjacent to the shaft.

The pit-bottom layouts are quite different if the shaft is away from the main haulage road. With an output from one side only those types shown in Fig. 359 (h) are possible; of these two systems, the layout with a reversing track is less expensive than that with the loop track. Where the output is drawn from both sides, the pit-bottom layouts shown in Fig. 359 (j) are recommended, as they are very efficient whatever the direction of the shaft axis. Both types have the disadvantage of a long travel for the locomotive to get from the full to the empty side; this difficulty, however, may be overcome by a special track for the locomotive to pass by the shaft, which results, however, in a greater width for the pit bottom, and track crossings are unavoidable at point A.

Track plans for the most frequent types of pit-bottom layouts are

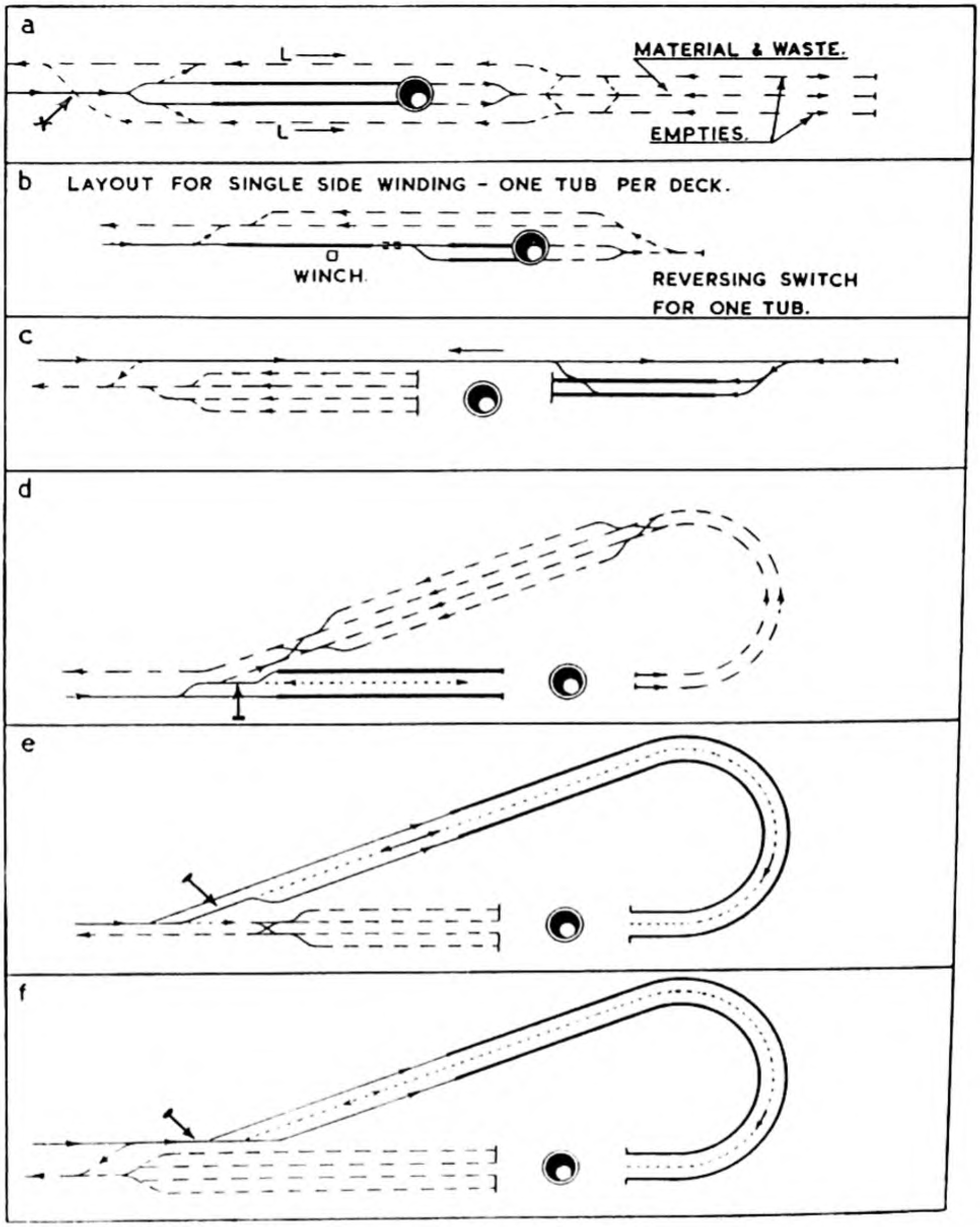


Fig. 360

shown in Fig. 360, a to m; for clarity, only the most important switches are marked.

(*d*) *Constructing a gradient plan.* When elaborating a levelling plan for a pit bottom, a base has first to be determined, and generally the centre of motion of the full-side tilting platform is chosen. The height of the top of rail above this point is marked zero, and al-

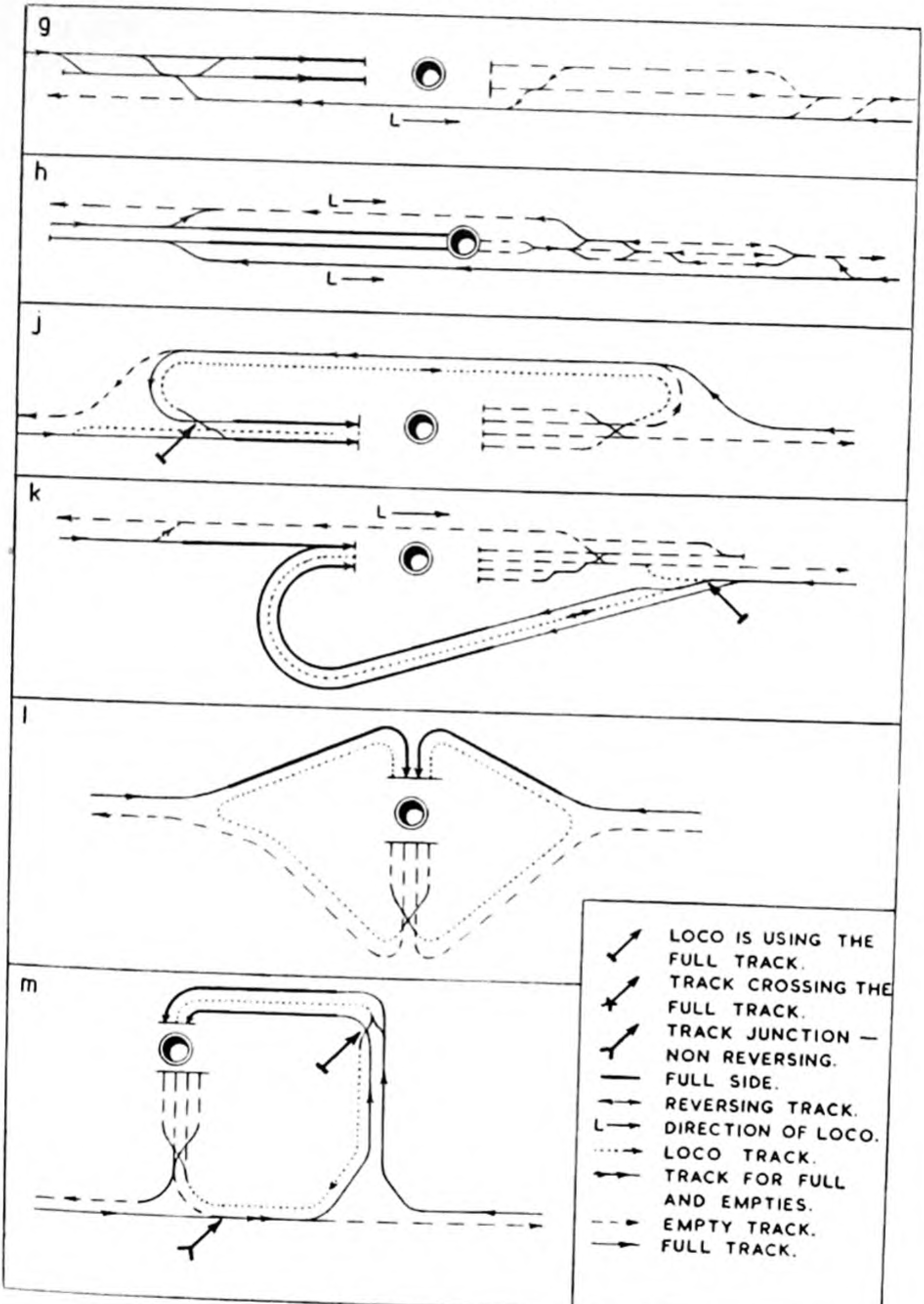


FIG. 360 *(continued)*

calculations are based on this datum. Starting from this point, the inclined tracks are drawn in either direction, the gradient being chosen with regard to the weight of the tubs, the friction of the

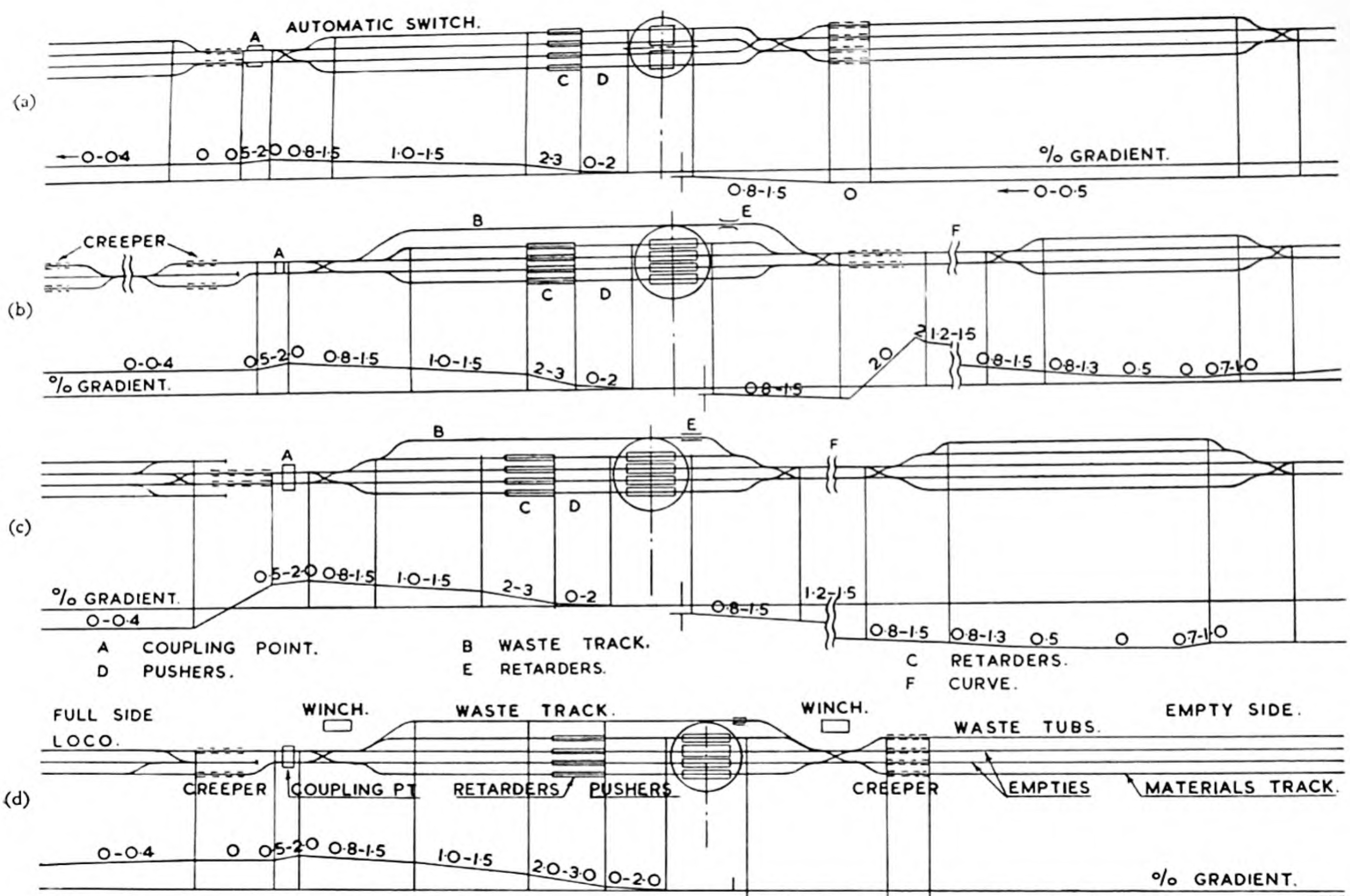
bearings and the track conditions. An example is shown in Fig. 361.

There is a drop in level between the entrance to the full siding and the end of the empty siding, depending on the gradient and the length of the track on which the tubs run by gravity. Where tub circulation within the pit bottom is accomplished, as far as possible by mechanical devices, inclined tracks are to be found only between the coal supply, chain-creeper on the full siding and the pushing chain-creeper on the empty siding. By limiting the length of inclined tracks as much as possible, the loss of level is not more than about 3 feet in the case of a single winding system with a cage capacity of eight tubs. In the case of shafts with a double winding system, it is possible to reduce the reduction in level to not more than 2 feet 6 inches because fewer switches are necessary. For smoothly running large mine cars and winding four cars per cage, the amount is still less. If the tubs run by gravity, the drop on the empty side depends on the length of the pit bottom; for small tubs it ranges from 1 foot 8 inches to 2 feet 6 inches. Curves in the form of a semicircle having a radius of 45 feet and a gradient of 1·4 per cent. give a reduction in level of the order of 2 feet. With a radius of 60 feet, however, the drop is almost 3 feet, which means that it is equivalent to the reduction in level up to the shaft. On the full side, tubs running by gravity drop almost 3 feet, providing the length of the pit bottom is sufficient for a whole train of small tubs to be collected on the same track.

It is obvious that the difference in level between the entrance to the full sidings and the end of the empty sidings must be compensated for by locomotive haulage, as the empty trains have to reach the main haulage road beyond the entry to the full siding. Due to the longer distance, it is easier to make up for this difference in level in a loop than within the pit bottom and in any case the track or the loop must rise. The maximum gradient admissible for locomotive haulage is 0·5 per cent. on the straight and 0·3 per cent. in curves.

As previously mentioned, the loss of level is rather small if the pit-bottom haulage is effected by mechanical devices. It may be reduced still more by laying the track slightly to the rise. Where gravity haulage results in a considerable loss in level, the use of a lifting chain-creeper is generally unavoidable and it is most frequently installed on the empty siding.

A number of levelling plans are shown in Fig. 361, a to d.


AUTOMATIC SWITCH.
A
C
D
(a)
← O-O·4
O
O5-2O
O·8-1·5
1·O-1·5
2·3
O-2
% GRADIENT.
O·8-1·5
O
← O-O·5
CREEPER
B
E
F
(b)
O-O·4
% GRADIENT.
2·O
1·2-1·5
O·8-1·3
O·5
O·7-1·O
(c)
A COUPLING POINT.
D PUSHERS.
B WASTE TRACK.
E RETARDERS.
C RETARDERS.
F CURVE.
FULL SIDE
LOCO.
WINCH.
WASTE TRACK.
EMPTY SIDE.
WASTE TUBS.
CREEPER
COUPLING PT
RETARDERS
PUSHERS
EMPTIES
MATERIALS TRACK.
(d)
2·O-3·O
O-2·O
FIG. 361

Section 5. General Layout for Skip Winding

In general, the pit-bottom layout is the same for cage and skip winding, the tippler acting as the main factor in the same way as the shaft axis in cage winding. Due to the different methods possible in placing the tippler, the pit-bottom layout for skip-winding shafts allows a wide range of variations.

An essential difference in the comparison with cage-winding shafts lies in the fact that pit bottoms for skip winding may be far narrower, since no tracks are needed for rubbish or materials. Because skip winding operates more regularly than cage winding and is not interrupted by man-riding, the storage capacity of a pit bottom for skip shafts need not be so high as for a cage-winding system and is generally shorter. Coal may be stored either in bunkers or in mine cars. Generally, mine cars are preferred for this purpose as high-capacity bunkers are rather expensive and cause degradation. With skip as well as cage winding, the section of the pit bottom depends to some degree on the intensity of the ventilation current.

The transport of cars up to the tippler may be the same as with cage winding. It is not necessary to distribute the cars among the different tipplers, each of them having a collecting track of its own. The tubs are separated by single or double locks. Generally, the tubs are pushed into the tipplers by pushers. Both in front of and within the tippler, the cars run against a gradient of 3 per cent., allowing the very useful advantage of the use of semi-automatic tipplers with a light, spring-type, non-return stop. The car is pushed into the tippler slightly more than the intended position, and as soon as the pusher is retracted the car slowly runs back by gravity and is held in the tipping position by the stop. Where there is a weigh-bridge in front of the tippler, this also is equipped with a non-return stop. The cars are first pushed over the weigh-bridge, on the next stroke they are carried forward and into the tippler, and when the next cars arrive they are pushed out.

Immediately behind the tippler there is an inclined track on which the cars run down to the empty siding. Chain-creepers should be the only haulage system on the empty siding if possible. Should there be sufficient space and the cars have not to pass over switches, the pushing chain-creeper may begin 6 feet behind the tippler; in other

cases, the tubs must run over a longer distance by gravity, particularly if they have to pass by the shaft.

With a suitable pit-bottom layout and tippler design, as shown in Fig. 362, the loss of level in the track layout resulting from the short slope behind the tippler is quite small, the rise in level is much greater. Under less favourable conditions, the drop in level due to the tipplers is much less than in pit bottoms for cage winding. In any case, it is easy to make up for this loss in level with smoothly ascending roadways.

If the haulage is exclusively by conveyors, the winding capacity of the shaft must correspond to the maximum performance of the con-

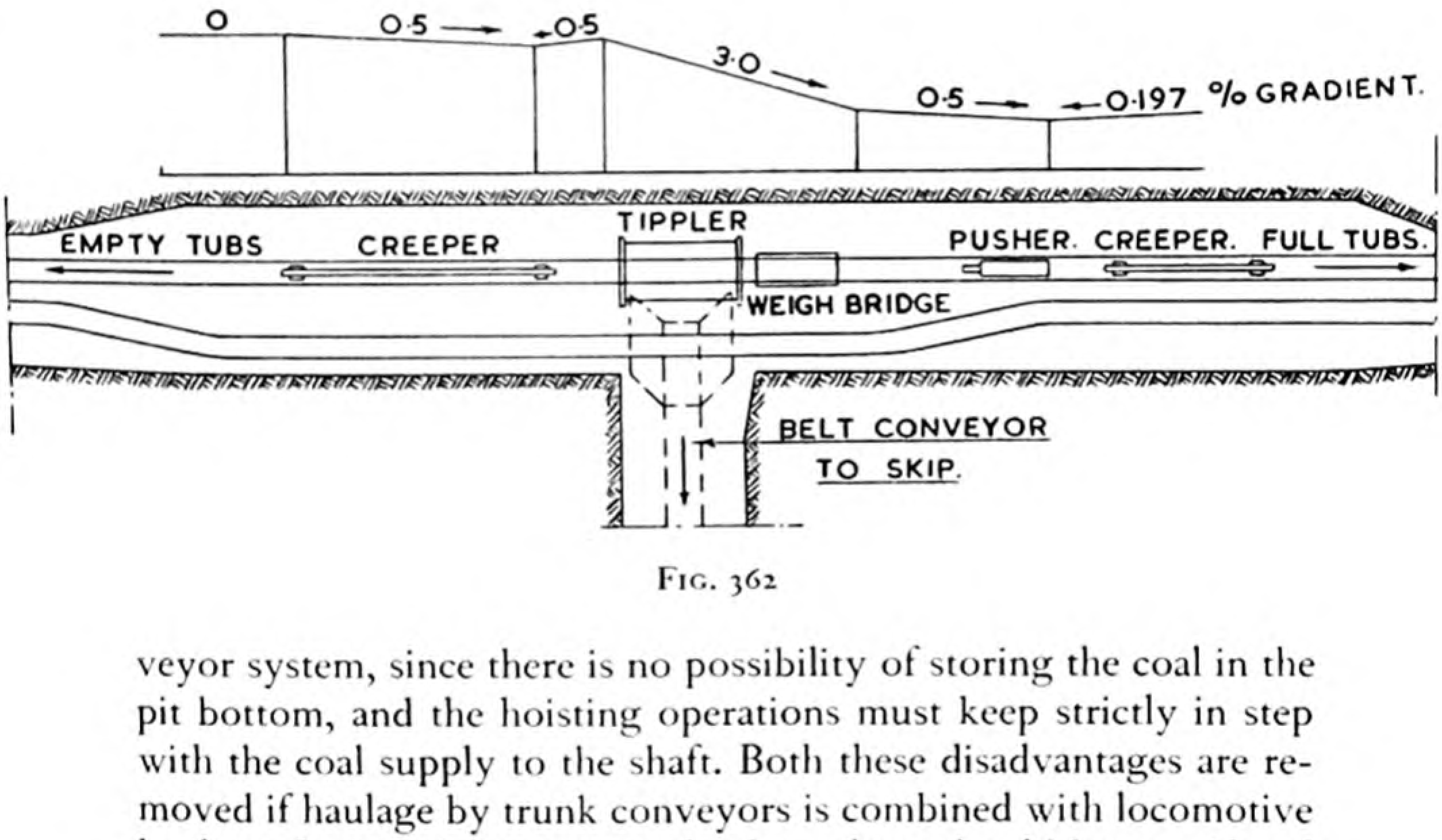


Fig. 362

veyor system, since there is no possibility of storing the coal in the pit bottom, and the hoisting operations must keep strictly in step with the coal supply to the shaft. Both these disadvantages are removed if haulage by trunk conveyors is combined with locomotive haulage. In practice, conveyor haulage alone should be considered only for short distances, e.g. in working out the shaft safety pillar.

(*a*) *Bunkers and filling pockets.* A separate filling bunker with a storing capacity equal to the winding capacity of the skip is necessary for each skip compact unit. The bottom of these bunkers are inclined at about 43 degrees and the coal slides down automatically.

A collecting conveyor is sometimes necessary between the tipplers and the filling bunkers as illustrated by Fig. 363, the speed of this conveyor being about 3 inches per minute. When running into the bunker the coal does not fall but slides.

Where the bunkers are installed immediately beneath the tipplers, they are fitted at the top with a compressed-air-operated distributing flap. The purpose of this flap is to close the bunker which has been filled or is discharging its contents, so that the second bunker can be filled. When the bunkers are fed from conveyors, the distributing flap is often replaced by a horizontal slide valve.

To prevent the coal from being thrown down into the bunkers, they are fitted with interior anti-breakable devices consisting of a

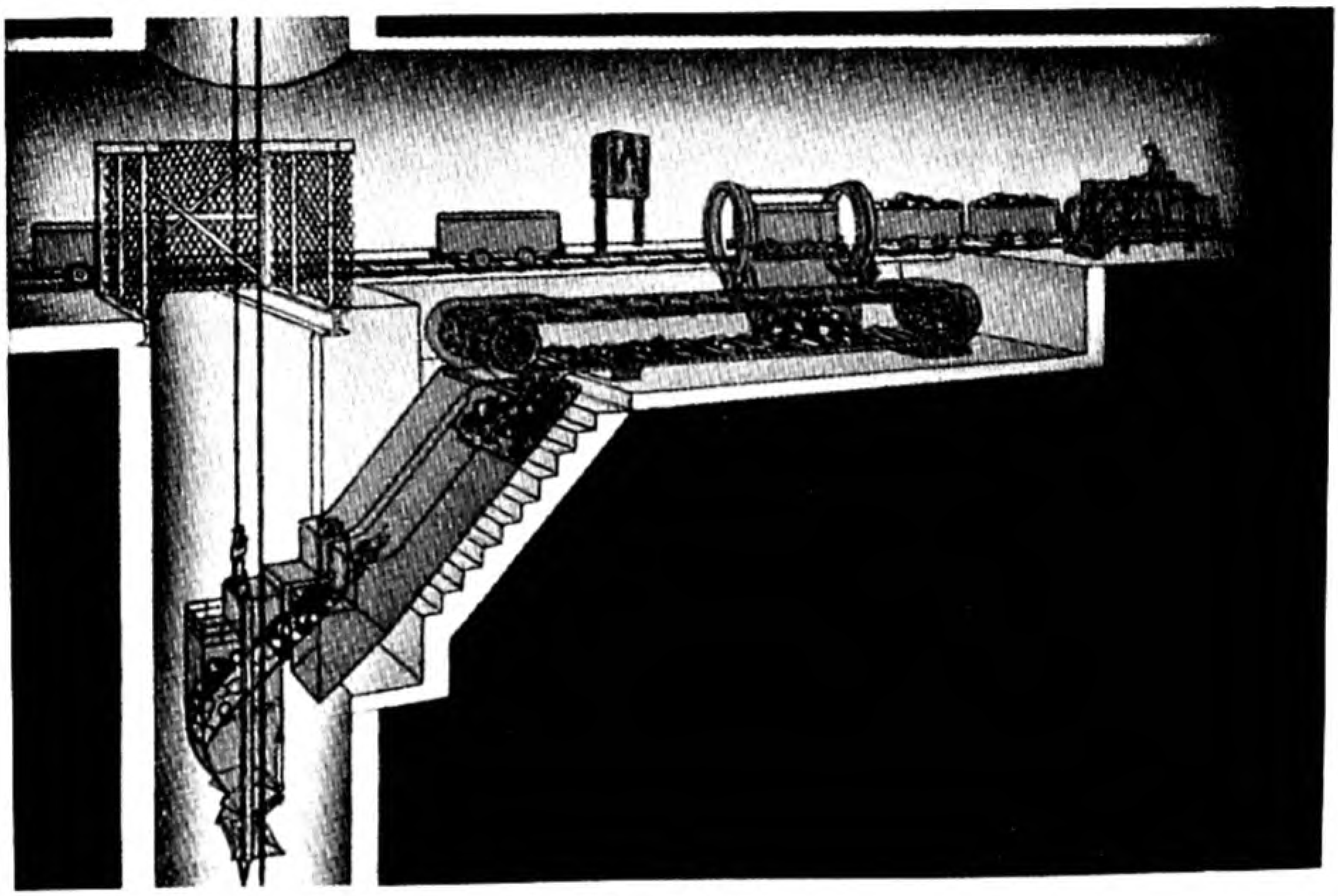


Fig. 363

descending plate covering the whole section of the bunker. The plate is fastened to an endless chain and slides between outside guide rails on the top of the bunker, as shown in Fig. 364. The motion of the plate is controlled by the tippler, going down by steps, the number of steps corresponding to the number of revolutions of the tippler which are necessary for the bunker to be filled, and in this way the coal level is always kept constant. After the last revolution of the tippler, the descending plate reaches its final position and makes way for the whole of the coal in the bunker to slide down and gather in front of the outlet door. When the bunker is fed from a trunk conveyor or the collecting conveyor, previously mentioned, the descending plate is controlled by a regulator which is tripped

whenever a small quantity of coal has gathered at the door of the bunker. With this system of control, the plate descends in many steps.

The descending plate is generally operated by a reversible electric motor which allows the plate to slide down slowly and to return

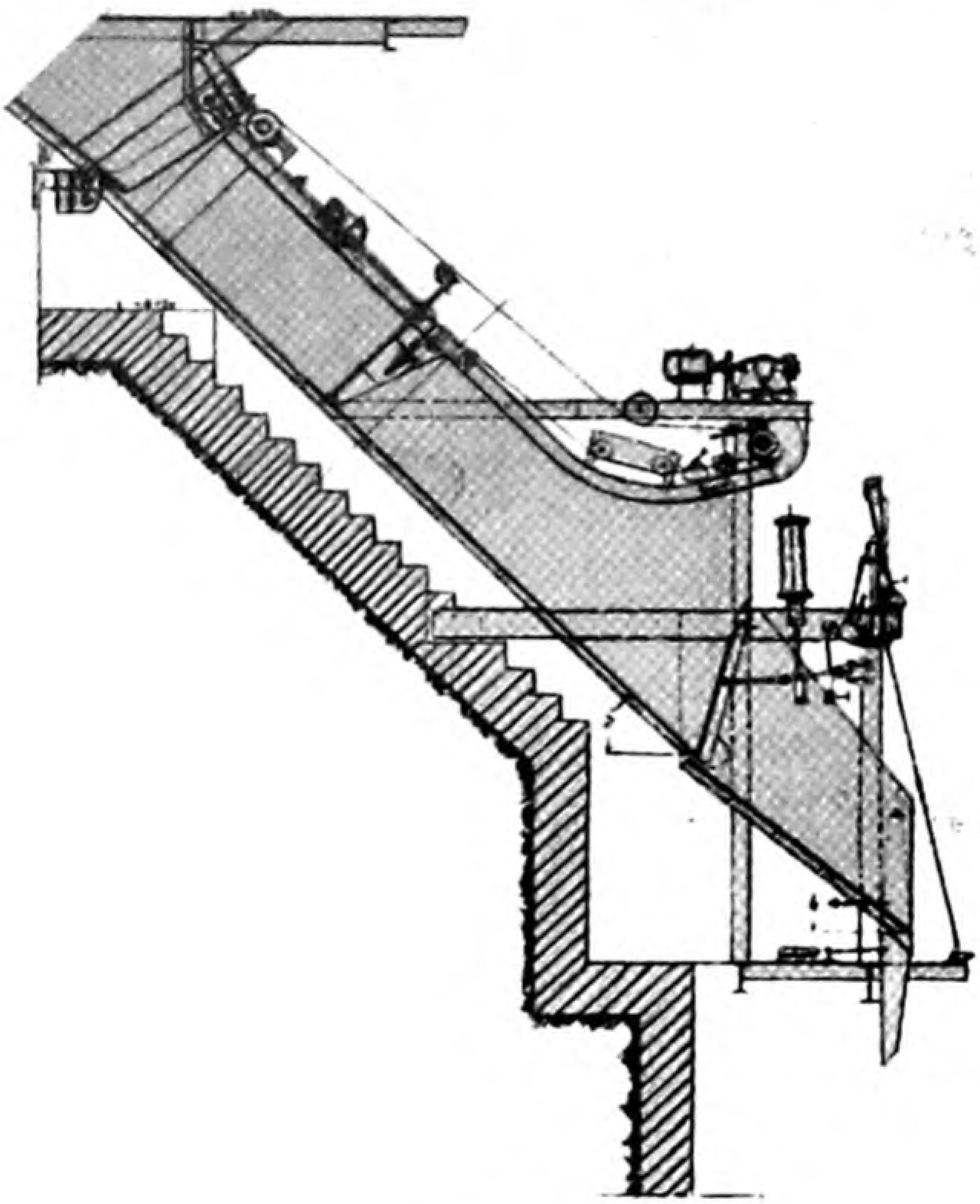


FIG. 364

quickly, the plate being stopped by a loaded brake. In the case of a breakdown, it is possible to move the descending plate by hand into its bottom position in order to operate the filling plate without the anti-breakage device.

A pocket valve operated by compressed air serves as an outlet from the bunker towards the shaft, the coal running into the skip through connecting chutes.

(*b*) *Operation of filling equipment and interlocking devices.* The skip-filling plant is operated either by compressed air or electrically

by one man on the operating platform near the tippler. The mechanical devices for the filling plant are operated partly by hand and partly automatically. The pushers and the tipplers must be switched on each time. The reversing flap is turned automatically when the descending plate of the full bunker has reached its bottom position, while the plate of the bunker to be filled has reached its top position. The descent of the plate is automatically controlled by the revolutions of the tippler. The plate begins to ascend as soon as the top flap is closed, the travel of the descending plate being limited in either direction by special switches. The outlet of the bunker and the filling chute are operated by the same lever, the chute being tripped at a higher speed.

A number of interlocking devices secure the right sequence of the different operations, thus preventing accidents. The pusher cannot be operated unless the tippler is ready to receive the cars; on the other hand, it is impossible to turn the tippler when the cars are being loaded. When the bunker is full, the tippler is blocked and the collecting conveyor is stopped in case the reversing flap should not be closed. Similarly, it is impossible to turn the tippler if both bunkers are full or if the outlet from the bunker to be filled is open. The outlet valve cannot be operated unless the bunker is completely filled and the skip is in the filling position.

Regular operation of the filling plant is controlled by a luminous control board in front of the operator's platform. On this board the position of the reversing flap, the process of filling the bunkers according to the position of the descending anti-breakage plates, opening and closing of the bunker outlets, and the arrival and the direction of the travelling skips are represented by appropriate signs.

The interlocks, the automatic controllers and the light signals depend on a number of contacts which are operated by the different devices and the skip. When the skip is filled, this is automatically communicated to the winding engine by the closing of the outlet door.

(*c*) *Dust suppression*. Wherever coal is sliding or falling it gives rise to dust formation. In skip-filling plants the dust formation is high, especially where the mine cars are tipped and the skip is filled. In order to prevent the dust from penetrating into the pit bottom, it is necessary to enclose the tippler and to provide a seal between the tippler and the filling bunkers. These bunkers and the connection

between them and the skips must be sealed also. Finally, dust must be prevented from getting out of the skips being filled. One of the most serious problems is the difficulty in making a close connection between the filling bunker and the skip, since the skip is moving while being filled. Sealing brushes are often used, but in spite of these, dust finds a way out of the skip and it is advisable to seal the shaft.

No kind of sealing will prove completely efficient as long as there is a positive pressure in the sections to be sealed. Such a condition will occur in the skip, in particular during the filling process. A connection pipe between the skip and the upper part of the bunker will greatly assist in overcoming these difficulties.

It will depend to a great degree on the nature of the coal, whether or not the above measures are sufficient. As a rule, dust troubles are greater if the skip-winding system has been installed in the downcast shaft than in the upcast. In difficult cases a de-dusting plant must be installed, and the possibility of such an installation must be taken into account if the necessity should arise later on.

Section 6. Auxiliary Shaft-bottom Accommodation

When planning a pit bottom, regard should be had at the outset to the location of other rooms which eventually will be necessary. As far as possible, these future rooms should be at right angles to the strike; in flat formations they should be excavated in a resistant bed. Long and narrow rooms are preferable. To provide good ventilation, an appropriate distance between the new rooms and the roadways should be maintained. The rooms in question may be situated parallel to each other outside the pit-bottom area in the narrower sense of this term, or they may be excavated in series, parallel to a pit-bottom gallery. All rooms should be easily accessible, and in case of necessity it should be possible to enlarge them without difficulty.

It is necessary to have special regard to the drainage problem. On account of the shaft rising main, the pump station should be as near as possible to the shaft. It is good practice in the case of high quantities of water to separate the water standages into two and to drive them as straight as possible; overlying workings should not be undercut by the standages. The distance between the shafts and the sumps should be sufficient for the latter to be kept watertight.

It is good practice to drive the standages towards the influx of the water.

When selecting the site for standages and the transformer substation, regard should be made to the layout of the main cables.

It is useful to have the stores and the workshops adjacent to each other near the material-supply shaft. There must be a convenient connection track between these rooms and the pit bottom.

Locomotive sheds should be installed near the workshops; they must be connected with the empty side by a railway track.

Material stores and tool-repair shops should be easily accessible from the man-riding shaft.

A stone crusher may be installed either on the full or on the empty side of the pit bottom.

The operator of the pit-bottom telephone should be so placed as to have complete control of the whole haulage traffic.

CHAPTER 6

VENTILATION

PART I

GENERAL PRINCIPLES OF UNDERGROUND VENTILATION

Section 1. *General Introduction*

THE ventilation of mines has to fulfil the three main purposes of providing sufficient air for the respiration of men and animals underground, of diluting and removing noxious gases and of providing atmospheric conditions underground favourable to the workers. The latter purpose is particularly important in hot and deep mines.

The most important function of the ventilation system is the dilution of noxious gases to render them harmless and to remove them from the mine. The quantity of methane in the general body of the mine air is limited to a definite percentage by Regulation. In Britain the maximum is 1·5 per cent., in Germany 1 per cent. and in Czechoslavakia 2 per cent. The quantity of air to be circulated in the mine to restrict the presence of methane to these values depends upon the individual mine conditions, and generally varies between 100 cubic feet per minute and 140 cubic feet per minute per daily ton of output. A mine with an output of 1,000 tons per day will require a minimum of about 100,000 cubic feet per minute, and a mine drawing 4,000 tons per day not less than about 400,000 cubic feet per minute.

The ventilation of a mine is carried out by providing a continuous flow of air, which enters the mine through one or several shafts and leaves it through one or more shafts. The shafts are called upcast or downcast shafts depending upon the direction of the airflow. At small mines, two shafts are generally sufficient, whereas at larger mines, several downcast and upcast shafts may be used. In some cases where the seams being worked outcrop to the surface, drifts or inclined shafts may be used. The air current is directed round the mine workings through intake and return airways between the downcast and upcast shafts. The total quantity being circulated is divided into a number of parallel and series circuits, or

'splits', as required, in order to ventilate the individual districts in the mine as efficiently as possible. The intake quantity is the quantity passing between the downcast shaft and the face, while the return quantity is that which passes between the faces and the upcast shaft. In the conventional mining systems, the intake and return roadways are at the same horizon and driven in the seam. In horizon mining development, the lower intake and upper return levels are separated, connection between the two levels being made via the faces and staple shafts.

Section 2. Exhausting and Forcing Ventilation Systems

The air current in the mine may be produced by either an exhausting or forcing fan set at the surface at the upcast or downcast shaft respectively. With an exhausting fan, a pressure below that of the atmosphere is maintained in the upcast shaft, causing flow from the downcast, round the workings and through the upcast shaft to the surface. The forcing fan at the top of the downcast shaft produces a pressure above atmospheric pressure, thus forcing the air round the workings to the upcast shaft and to the surface.

Exhaust ventilation systems are almost universally used since ascensional ventilation can be adopted, the air travelling to the rise in inclined workings and towards the return airway situated at the higher level. The air-flow is therefore in the first instance drawn into the deepest part of the mine, which is generally the main shaft bottom and level haulage road. With an exhaust ventilation system, the main shaft becomes the downcast and thus there is no interference with the winding due to the necessity for air-locks. Ascensional ventilation takes advantage of the increase in the air temperature and consequent reduction in density, as the air circulates the workings towards the upcast shaft. The effect of moisture underground also assists in ascensional ventilation. It is often assumed that methane can be more easily removed by an ascensional than a descensional system. This assumption has been refuted by Wijffels, who showed that methane, which quickly diffuses in air, can be moved from points of lower pressure to points of high pressure, i.e. removed by an air current in a direction contrary to its natural flow.

Descensional ventilation has been practised successfully in some cases, either throughout a mine or for individual sections. Descensional ventilation has many advantages. The formation of a dust-

laden atmosphere is less likely where the air moves in the same direction as the coal. It is important to note that with descensional ventilation, the fresh air is first brought down to the upper level and has not been warmed up to the same extent as in ascensional ventilation, where the air travels along the deeper haulage roads before reaching the face. Where the horizon interval is between 150 and 200 yards, the strata at the lower level is from 4 to 6 degrees C. higher in temperature than at the upper level, depending upon the geothermic gradient. With descensional ventilation for the whole mine, it is thus possible to bring cool air direct to the face, and a forcing fan can be installed at the subsidiary shaft so that this becomes the downcast shaft, which is sunk only to the upper level.

The exhaust ventilation system applied to horizon mining development in which separate intake and return airways at different levels are maintained is shown in Fig. 365. The depression created by the fan at the upcast shaft is 8 inches water gauge (W.G.), while the pressure at the downcast is zero (atmospheric), and the depression gradually increases up to the maximum at the upcast shaft top in the direction of the air current. The ventilation pressure is absorbed according to the resistance of the individual sections of the airway. In the example, the pressure drop in the downcast shaft is $2\frac{1}{2}$ inches, the upcast shaft $1\frac{1}{2}$ inches (both shafts absorbing half the available pressure), the intake and return levels each absorb $1\frac{1}{2}$ inches, and the face 1 inch.

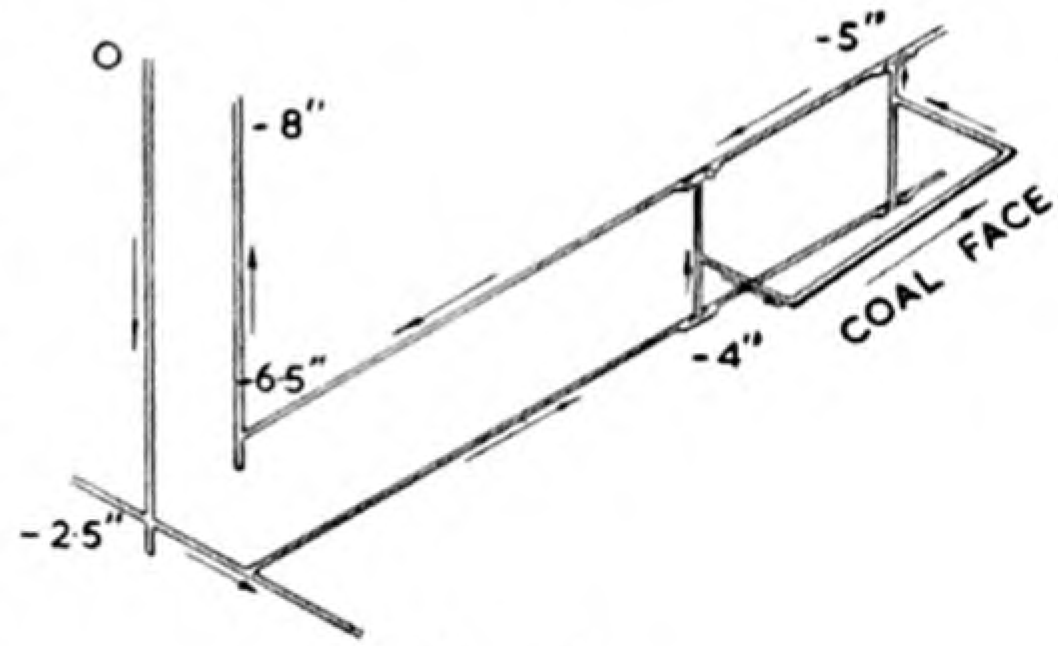
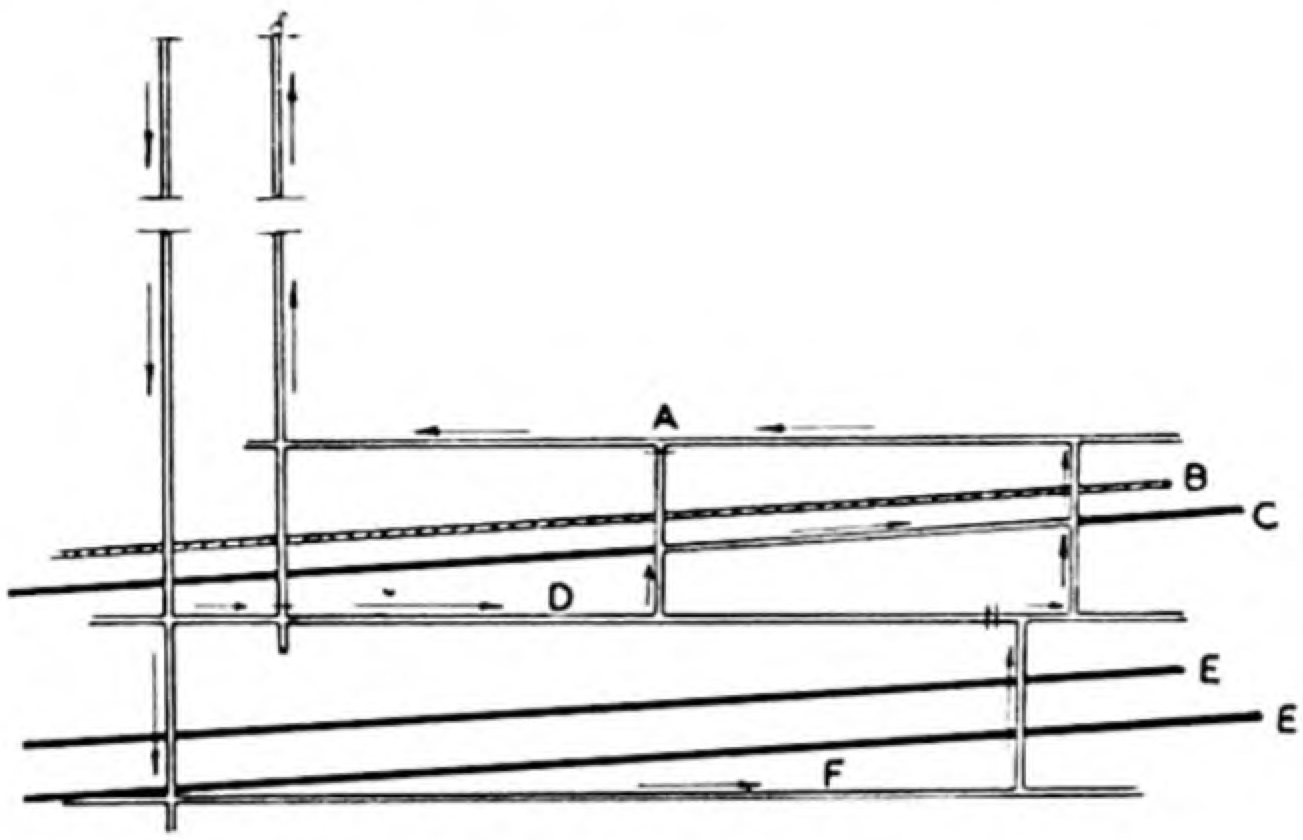


Fig. 365

In the above case it has been assumed that only one intake and one return level are available, a case which frequently arises. It is possible to have two intake levels and one or two return levels. This is usual when the new lower haulage and intake level is being driven and the old haulage road is still in operation. The intake air entering the new horizon is brought to the old main haulage level through one or several staple shafts, part of the level being used as a return to the upcast shaft.

This system is possible only if the upcast shaft has been sunk to the haulage horizon; if this is not the case, the return air must be brought to the main return horizon through staple shafts or rises and then to the upcast shaft as shown in Fig. 366.



A. RETURN AIR HORIZON.
B. SEAM WORKED OUT.
C. SEAM IN OPERATION.
D. INTAKE AIR HORIZON. (MAIN HAULAGE HORIZON.)
E. VIRGIN SEAMS.
F. INTAKE AIR HORIZON. (IN DEVELOPMENT.)

Fig. 366

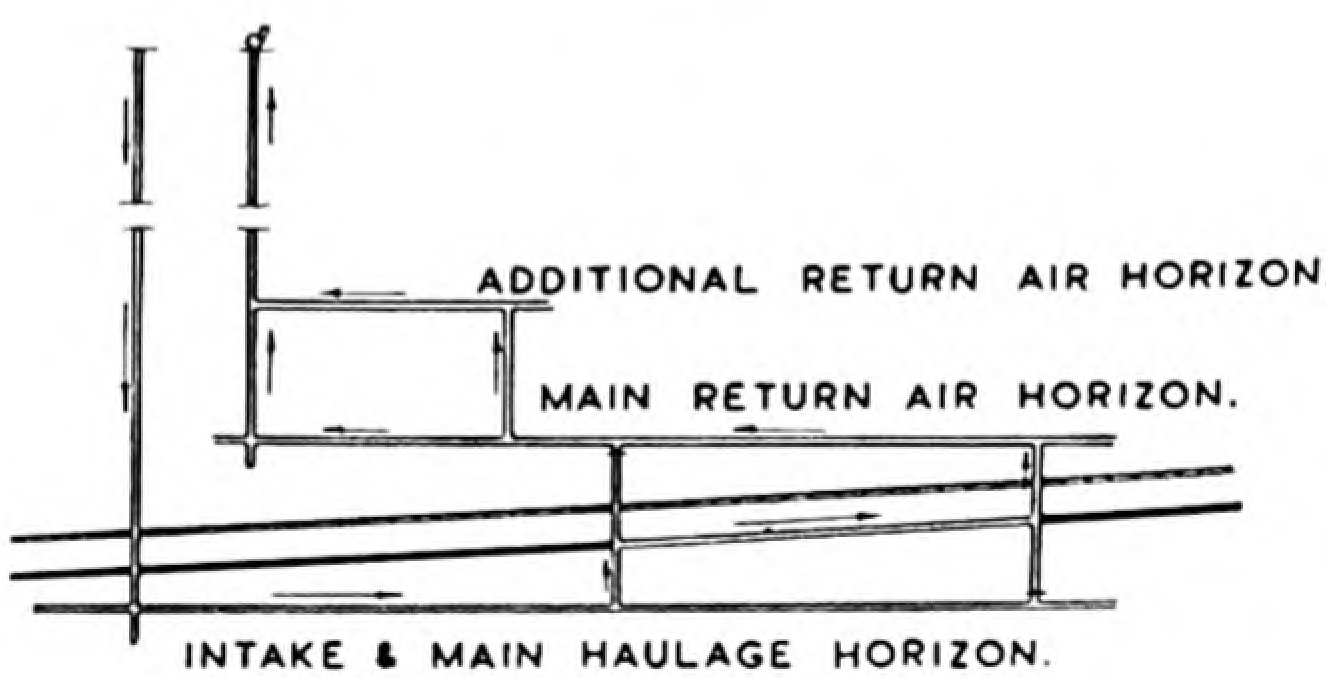


Fig. 367

Parts of an older return level above the main return level may be used as a return, either to supplement the main return level or when reserves still remaining above the main return level are being worked. The splitting of the main return into parallel circuits reduces the overall resistance, thus saving power, *vide* Fig. 367. Refer to Section 4.

Section 3. The Number and Location of the Shafts

Where there is the minimum requirement of only two shafts, one shaft will be the main winding and downcast shaft and the other the upcast. The downcast shaft is generally situated in the centre of the royalty or coal reserves, but it may be nearer to one or other of the boundaries, depending upon the distribution of the major portion of the reserves if they are steep and are dislocated due to faulting.

The upcast shaft may be located near the downcast shaft, since in this position it can serve as a reserve winding shaft using the same surface installations. When the shafts are adjacent to one another, the intake air must travel a certain distance from the downcast shaft

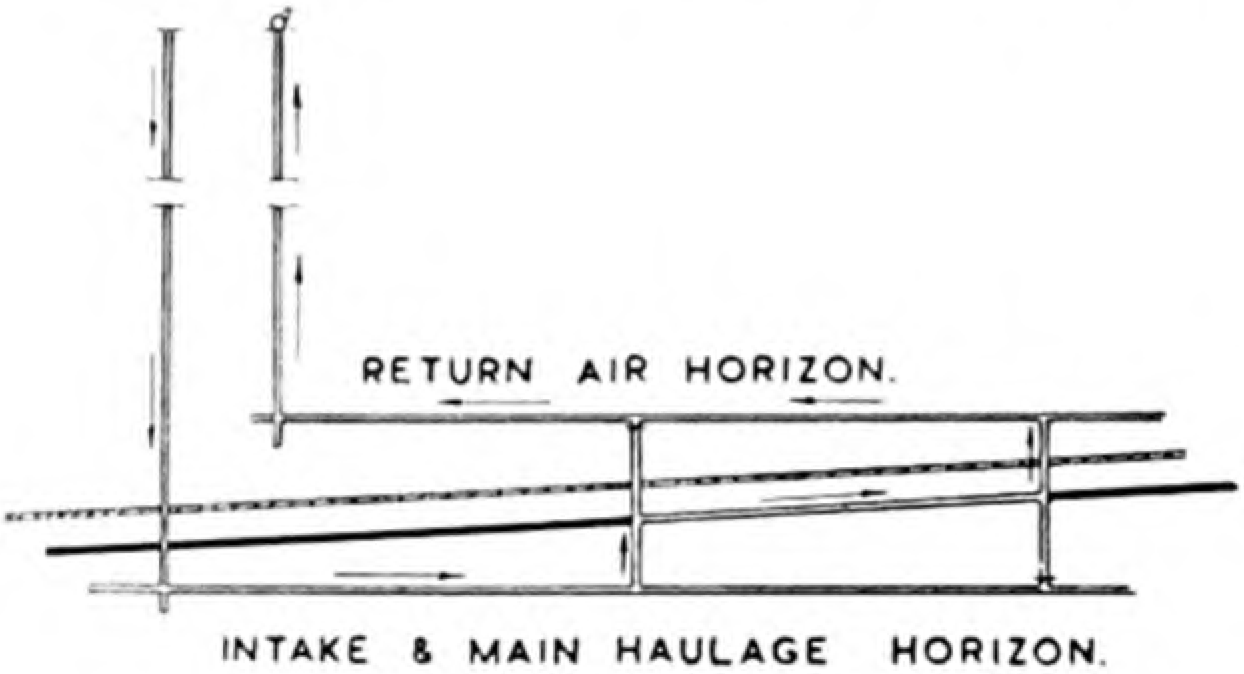


FIG. 368

and the same distance back as return air to the upcast shaft. In German mining practice this system is called 'retrograde ventilation', *vide* Fig. 368.

During the course of mine development, the retrograde system is probably the best. When the shafts have reached their respective levels of the main haulage and return air horizon, and drivage of these levels has commenced, it is easy to connect the two levels by staple shafts, thus providing immediate 'through' ventilation with short connections. The same advantage is gained when initially developing faces near the shafts. When the upcast shaft is some distance from the downcast, a greater time must elapse before through ventilation can be achieved, since longer roads have to be driven for this purpose.

The farther the coal faces are from the shafts and the nearer they are to the boundaries, the more unfavourable will the retrograde system become. The distance which the air has to travel is doubled

so that the resistance and the power required to overcome it are correspondingly increased. Furthermore the inevitable short-circuit leakage which occurs is a major disadvantage of the retrograde system in which the intake and return air will usually follow parallel roads. The system therefore should be limited in application to within a certain distance from the shafts. If the shaft is near the royalty boundary, the maximum distance travelled by the air would be limited to from 2,000 to 3,000 yards. The air would then travel from the main shaft in the centre of the royalty to one or several upcast shafts near the boundaries. This system has been termed 'boundary ventilation' and is illustrated in Fig. 369.

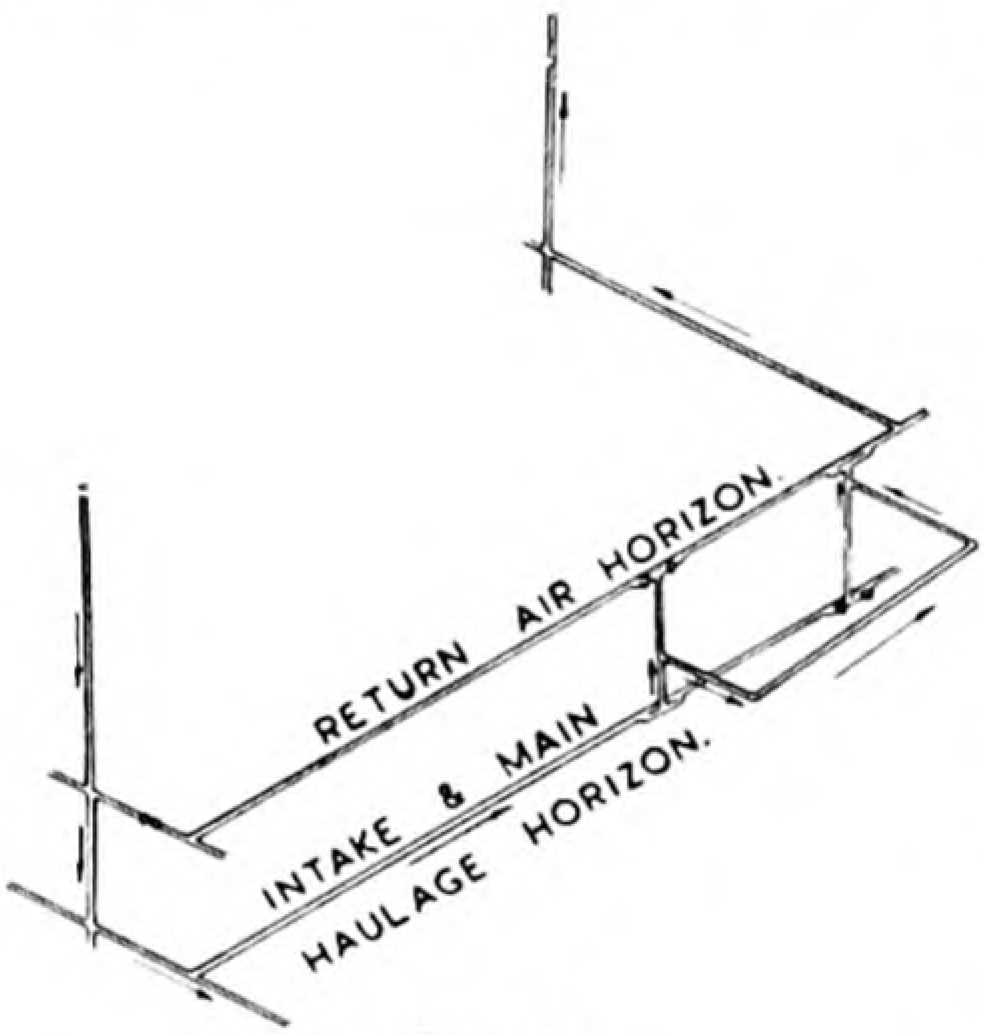


FIG. 369

The shaft layout may be reversed with the downcast at the edge of the royalty and the upcast located centrally. In this case the system has been called 'middle ventilation', the layout being shown in Fig. 370.

Where only two shafts are in use, the boundary ventilation system is seldom used. The conditions are entirely different where more than two shafts have been sunk. The necessity for three, four or more shafts, which can be adopted to give the advantages of boundary or middle ventilation, is also dependent upon the quantity of air which has to be circulated. One intake or return shaft may be insufficient to pass all the air considered necessary. This factor is specially important when there are long airways in the system, absorbing a large proportion of the available water gauge and providing serious quantity losses due to leakage.

The general rule in view of these factors is to provide at least two adjacent shafts in the middle of the royalty area and, as shown in Fig. 370, to sink one or more additional shafts near the boundaries as required. This development covers a long period, and in the course of time such a mine has several ventilation systems. In the case shown in Fig. 370 these are: the retrograde system connected

to the central, main downcast and upcast shafts, one boundary system with the main downcast shaft and a third boundary upcast shaft and, finally, a middle system with a boundary downcast shaft passing air to the central upcast shaft. The shafts may be utilised in other systems as the mine develops. Thus, it may be suitable to use the central shafts as downcast shafts and the two boundary shafts

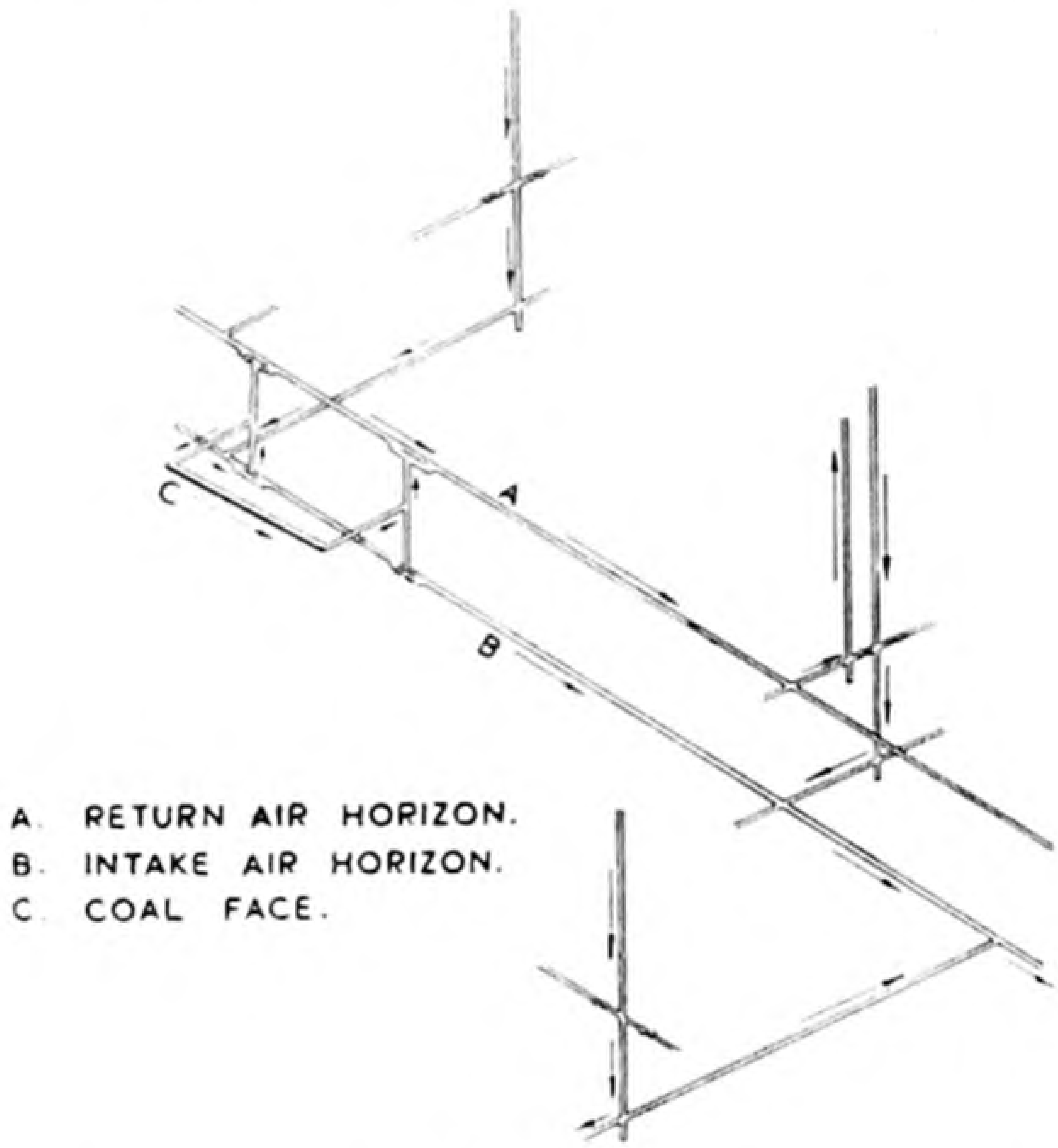


FIG. 370

both as upcast shafts. There can be no fixed rules on what constitutes the best system, the only guide is safety and economy in ventilation.

If the three systems described are examined critically, it should be noted that the possibility of short-circuiting or leakage is greatest with the retrograde system. In this system the distance between the intake and the return airways is usually small and, since the pressure difference between the two shafts bottoms is large, leakage can readily occur at this point. The result of this considerable leakage from the intake to the return airway reduces the quantity delivered to the working faces. It should be noted that the adoption of middle or boundary ventilation systems makes it easier to use parts of the same level for intake or return airways, without the need to make and maintain parallel roads.

The boundary shafts may be used to serve as ventilation shafts and for man-riding and transport of materials. The advantage of shorter travelling distances for men and materials is obvious. Mines having two main shafts and several outside shafts have been called 'combined mines'.

Section 4. The Division of the Total Quantity of Air by Splitting

The essential purpose of mine ventilation is to supply fresh air in adequate quantities to the coal faces in development and in full operation. This can be realised only by dividing the air current into

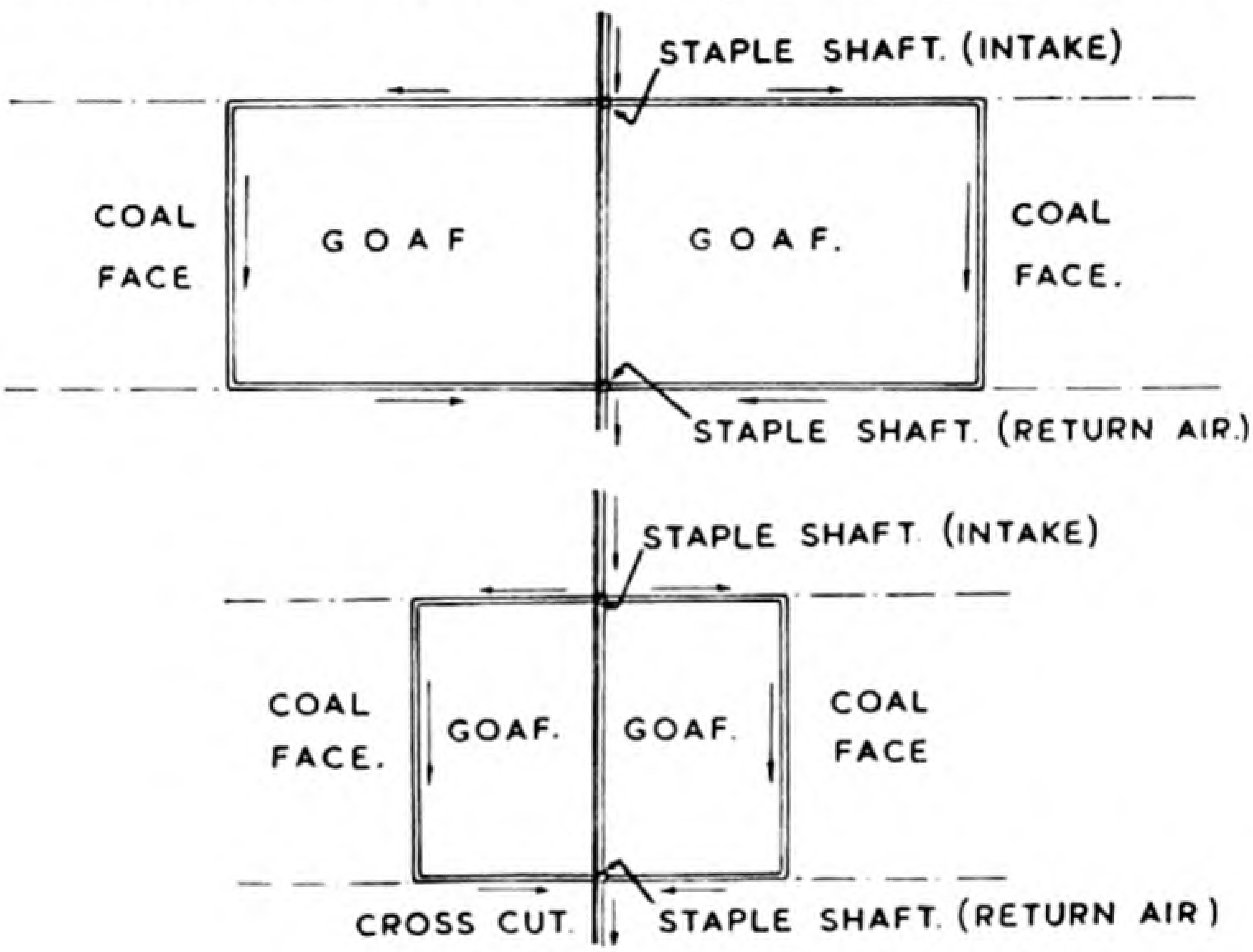


FIG. 371

a number of splits. If there is more than one downcast shaft, the first division is already taking place at the surface. Where there is only one downcast shaft, division starts at the pit-bottom or at the junction of the pit-bottom with the laterals, from which cross-cuts are branching off with an air split for each of them. These splits, which are further split into as many branches as coal faces, are connected to the district cross-cut. The air will reach the faces directly through the gate roads, or indirectly through the staple shafts from which it enters the gate roads.

From the coal faces the air is conducted through return gate roads and thence to the return air horizon. This can be done through the upper part of staple shafts or through rises to the return district cross-cuts.

Headings, gate roads and staple shafts also get fresh splits. The same applies to workshops and engine houses, while further quantities of fresh air are taken for the development headings in stone and coal. The air returning to the intake from development headings in coal has to be adequately diluted before reaching the gate roads leading to the faces.

The German practice is to limit the number of men at work in any single split or 'air panel', usually comprising one or more faces and the associated gate roads, rises and staple shafts. The number of men in each split is limited to 130 in non-gassy mines and 100 in gassy mines. According to German regulations, two neighbouring faces, as shown in Fig. 371, are considered to belong to the same air circuit or split, even if they are being ventilated separately, as long as the distance between them is no greater than 100 metres.

The division of the fresh air into a considerable number of splits is not only necessary to supply all parts of the mine with the required quantity of fresh air, but also to keep the speed of the air in the roads within reasonable limits. In most cases the speed does not exceed 12 feet per second in conveyor roads and 17 feet per second in main haulage roads. In many countries a maximum limit has been fixed by the mining authorities. In Germany this maximum is 24 feet per second.

The result of intensive splitting of the air stream is not always favourable; this is especially so in hot mines. The reduced speed increases the time during which the air can take up heat from the surrounding strata and the intake air will have a higher temperature when reaching the face. In hot mines the number of splits should be kept to a minimum, which in each individual case is governed by the quantity of air, the velocity of the air and the location of the faces. Mines with high maintenance costs for roadways will also try to keep the number of splits down on the intake side. Usually the maintenance costs of the gate roads on the return side are lower and heating of the air on the return horizon is advantageous in increasing the natural flow, therefore intensive splitting on the return side is usually to be recommended.

The total quantity of air passing into the mine consists of what is needed for the faces, for other circuits and special purposes and to cover leakages. Leakages include short circuit quantities outside the actual air panels; these do not serve any useful purpose. The main

points at which leakages occur are at the upcast shaft structure and air-locks, and at the shaft bottom between downcast and upcast shafts, if these are adjacent. In the former case, the pressure difference is the total depression of the mine, and a considerable quantity is sucked in through small holes and crevices in the shaft cover and flows through the fan drift directly to the fan.

It is evident that the efficiency of the ventilation system will be higher if the quantity of air delivered to the faces is high and leakage quantities are kept to a minimum.

In the Ruhr, the average quantity of air reaching the air-panel is about 50 per cent. of the total quantity entering the mine, while 30 per cent. is allocated to other circuits and 20 per cent. to leakage. In favourable cases the quantity reaching the district split may be as much as 70 per cent., and in the worst cases it may be as low as 20 per cent. The quantities taken up by leakage and other circuits vary between 30 and 80 per cent.

It is of interest to note that only from 20 to 60 per cent. of the total quantity of air reaching the inbye district is passed to the coal faces, i.e. from 10 to 30 per cent. of the total quantity is utilised in ventilating staple shafts and rises or lost by leakage within the face area.

These approximate values can be related to the actual average conditions in the Ruhr, where the quantity of air required for a mine can be assumed to be about 110 cubic feet per minute per daily ton.

The following table illustrates the air distribution for an average mine where the ventilation has been reasonably planned and maintained:

	cu. ft. per min. per daily ton
Coal face	34
Staple shaft, rises, etc.	24
Other air circuits	28
Leakage at the surface fan drift	14
Leakage between the downcast and upcast shaft bottoms	10

It is usual in stating the total quantity of air passing in a mine to refer to the return quantity when making comparisons and relative measurements; the quantity of return air is preferable to the intake

quantity, since the return quantity can be fairly accurately measured in the fan drift and the total intake quantity can only be measured with difficulty at the pit bottom. It should also be remembered that the volume of air passing through the mine changes and that the capacity of the fan is dependent upon the return quantity.

Several factors influence the total volume of air passing in the mine. The air enters the downcast shaft and passes into the workings, where it is heated and expands, finally expanding again at the upcast shaft due to the reduction in pressure approaching the fan. An increase in volume is effected by the increase in temperature and also by the addition of moisture and of gases, such as methane and compressed air, where it is used for a power supply. Heating of the air from $50°$ to $75°$ F. would increase the volume by 5 per cent., absorption of water will normally account for an increase of 1 to 3 per cent.

The methane content of the return air in gassy mines may be about 0·5 per cent., while for mines using compressed-air power, the influence of the exhaust air on the total quantity passing in the mine is very important and can be an increase of as much as from 2 to 10 per cent. of the return quantity. In splits where numerous high-capacity pneumatic machines are at work, this percentage may be still greater during the actual coal-getting shift, especially if pneumatic stowing is being used.

The formation of carbon dioxide may also be mentioned. In numerous mines the CO_2 content of the return air may be as much as 0·5 per cent. The formation of CO_2 is usually accompanied by a reduction in the oxygen content, and in general calculations the increase in volume due to this gas may be neglected.

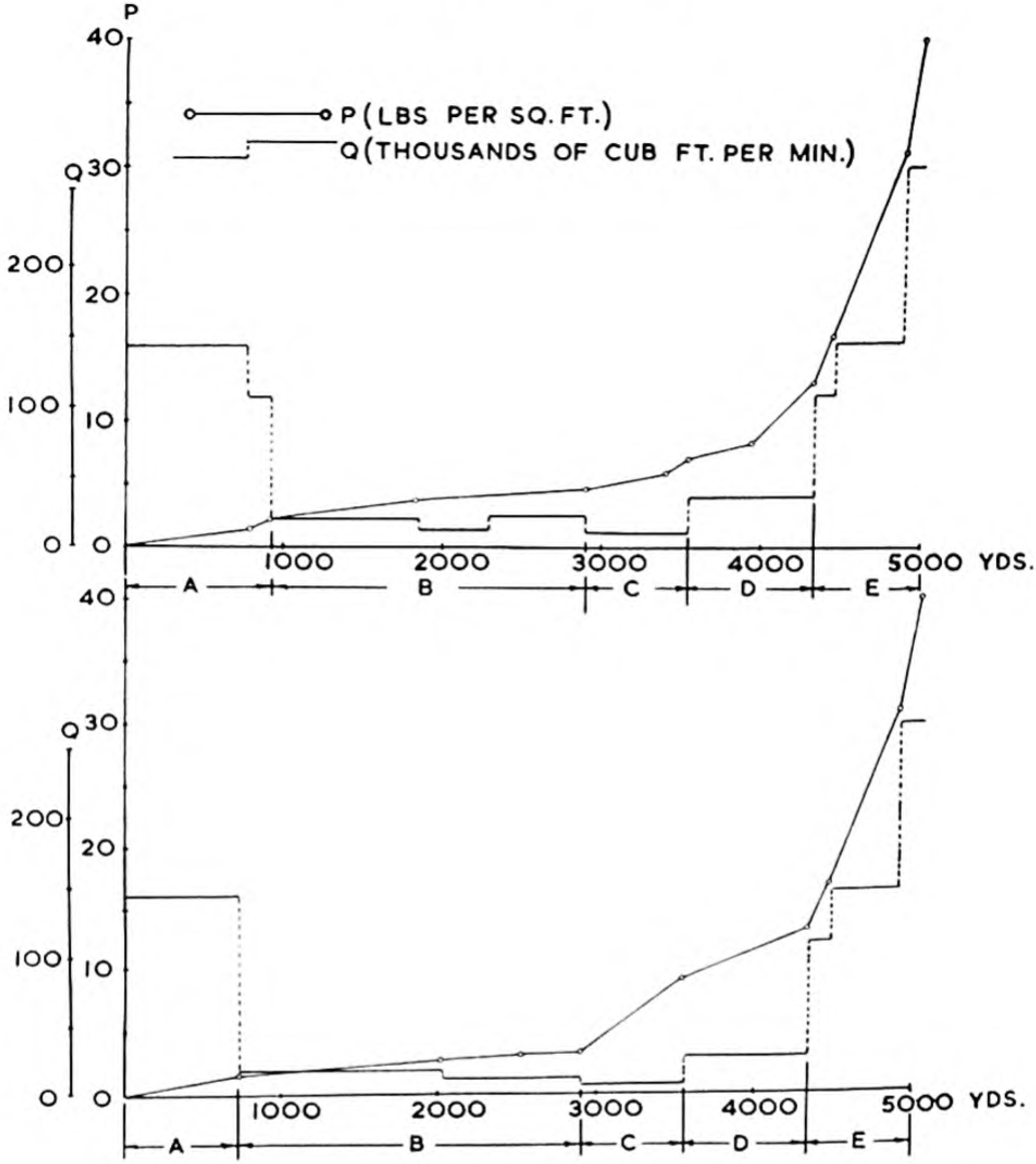
In general, the increase in volume caused by these various contributory factors in coal mines may range from 5 to 15 per cent., the use of compressed air being the most important item.

Section 5. The Division of the Total Ventilating Pressure

Detailed consideration of the pressure distribution throughout the mine is as important as the quantity distribution. The following sections of the ventilating circuit may be distinguished:

(*i*) the downcast shaft;

(*ii*) the level or ascending intake airways, such as main roads, staple shafts, rises and gate roads;



A. DOWNCAST SHAFT.
B. INTAKE AIRWAYS. MAIN HAULAGE HORIZON.
C. COAL FACE.
D. RETURN AIRWAYS.
E. UPCAST SHAFT.

FIG. 372

(*iii*) the coal face;

(*iv*) the level or ascending return airways, such as gate roads, staple shafts, rises and main roads on the return airway level;

(*v*) the upcast shaft;

(*vi*) the fan drift.

The average distribution of the total ventilating pressure in the Ruhr to these sections of the ventilation system are as follows:

	per cent.
Downcast shaft	20
Intake airways	16
Coal faces	10
Return airways	30
Upcast shaft	18
Fan drift	6
	100

From this distribution it can be seen that the major pressure loss is in the return airways, which require a total of 54 per cent. This proportion may be considerably higher in some mines. The return airways are probably too small in area to accommodate large quantities, and the air velocities are excessive. This loss can be reduced considerably by providing parallel airways. The upcast shaft may be too small in diameter and the fan drift layout may introduce high losses due to turbulent flow conditions. A high resistance in the fan drift can be caused by the throttling of the air-flow at the mouthing from the shaft into the drift. The deeper the shafts, the greater is the proportion of the available pressure which will be absorbed by them, and as the shafts will normally consume from 35 per cent. to 40 per cent. of the pressure, it is specially important to reduce the resistance of the shafts.

The graphs in Fig. 372 illustrate the pressure and quantity distribution in two splits in a mine in which the total number of splits are fourteen. The two circuits commence from the downcast shaft and are delivering air to other splits on the intake side and joining up with return air entering the return airway to the upcast shaft. It can be seen in Fig. 372 that a quantity of about 140,000 cubic feet per minute is entering from the downcast shaft, and that this quantity feeds splits of 26,500 and 3,500 cubic feet per minute and is reduced by about 10,500 cubic feet per minute before the coal face is reached. The return from the face is increased from another split by 25,000 cubic feet per minute to a total quantity of 35,000 cubic feet per minute, and the return finally joins other quantities in the upcast shaft, giving the combined return quantity.

The pressure graph shows the distribution of the total pressure drop through the whole system, the return airways absorbing a considerably higher proportion than the intake airways.

PART II

VENTILATION MEASUREMENTS

Section 1. *Quantity Measurements*

The general survey of the principles of underground ventilation shows the necessity of having a detailed knowledge of the pressure and quantity distribution in the mine as well as an appreciation of the atmospheric conditions of temperature and humidity. The measurements required to obtain sufficient information from which to compute these factors are equally important.

In order to calculate the quantity of air passing along an airway, the velocity of the air and the cross-sectional area of the airway must be measured. The quantity can then be expressed by the simple relation

$$Q = v\ a,$$

where Q is the quantity in cubic feet per minute, v is the air-velocity in feet per minute and a is the area of the cross-section of the airway in square feet.

The velocity of the air current is usually measured with an anemometer, but other instruments such as the Velometer or the Bochum velocity meter may be used to give a direct recording. The anemometer is a small air-motor geared to a series of dials which register the revolutions directly in feet. The reading is recorded over a definite time, usually not less than two minutes, and the air velocity computed in feet per minute. An instrument of the high-speed type is illustrated in Fig. 373. The measurement of low velocities may be carried out with the Ower low-speed anemometer shown in Fig. 374, in which the single dial registers revolutions which are converted to actual velocities by referring to a calibration graph for the instrument. Each anemometer should be calibrated before use and checked at intervals while in use. Individual instruments have different calibration curves, and for important ventilation surveys it is advisable to have the instrument previously calibrated by the National Physical Laboratory, who will provide the data upon which the calibration curve can be drawn. A typical

curve based on N.P.L. calibration is shown in Fig. 375.

FIG. 373

When taking velocity measurements with the anemometer, it is necessary to traverse the instrument vertically and horizontally over the section, thus covering the whole area of the roadway. The anemometer, which is usually a zero-setting instrument and is provided with a clutch-release pointer, should be attached to a stick so that readings can be taken as far from the observer as possible. The recordings should be repeated over several minutes and the mean observed velocity corrected to the actual velocity by the use of the calibration graph.

The Velometer has been developed to provide 'spot' readings from which the mean velocity over the section can be calculated. This instrument incorporates a swinging gate set on accurate pivots with its face between inlet and outlet orifices. The plate or gate is magnetically damped to avoid rapid fluctuations of the dial needle on the scale, thus allowing good readings over a low- and high-speed velocity range. Special orifice jets are provided for different

FIG. 374

ranges. The instrument, which is illustrated in Fig. 376, is also calibrated to read pressures, and can be used for pressure-difference measurements.

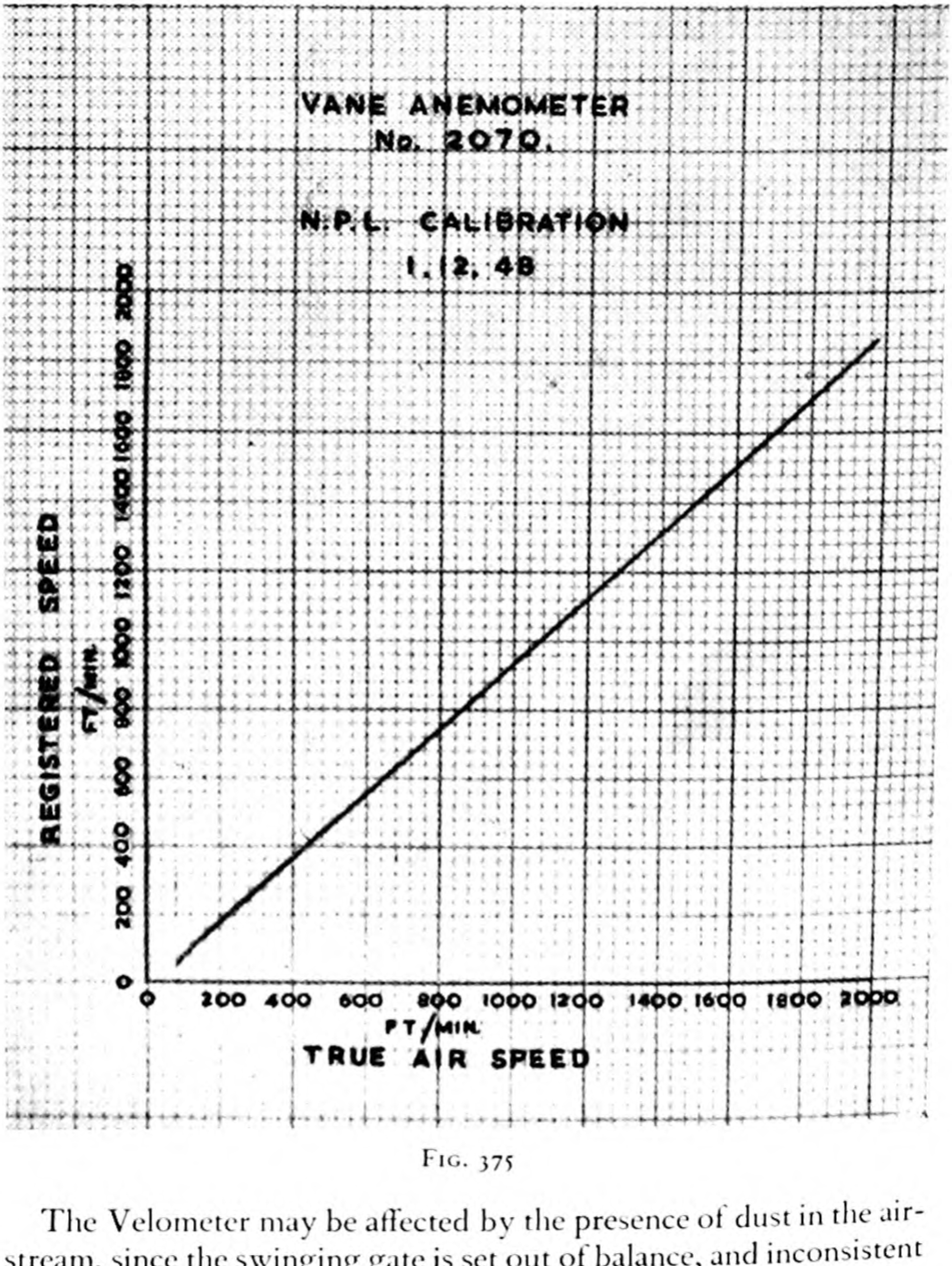


FIG. 375

The Velometer may be affected by the presence of dust in the air-stream, since the swinging gate is set out of balance, and inconsistent readings result. This is a serious disadvantage in general ventilation survey conditions, but the difficulty is overcome in the Bochum

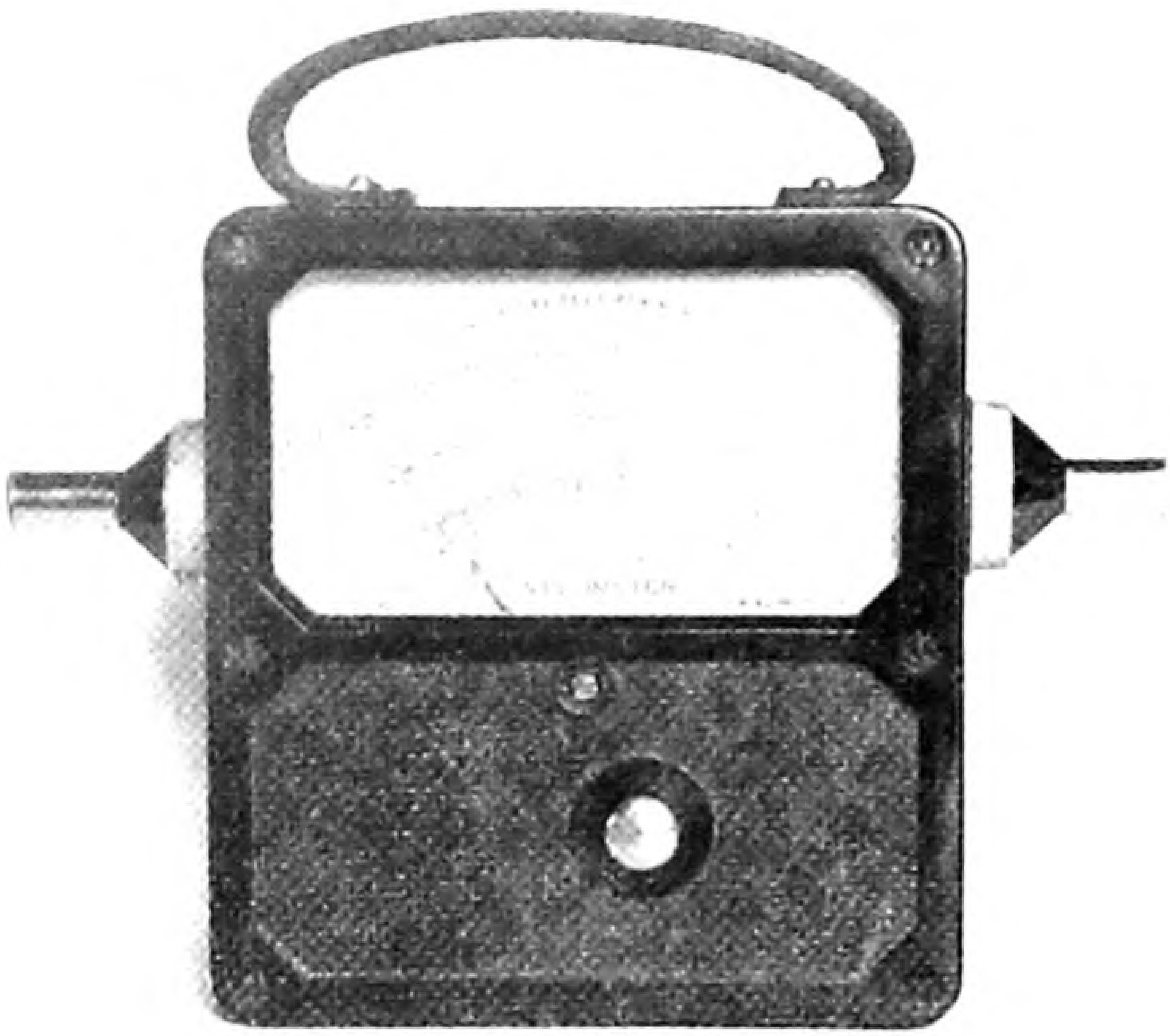


FIG. 376

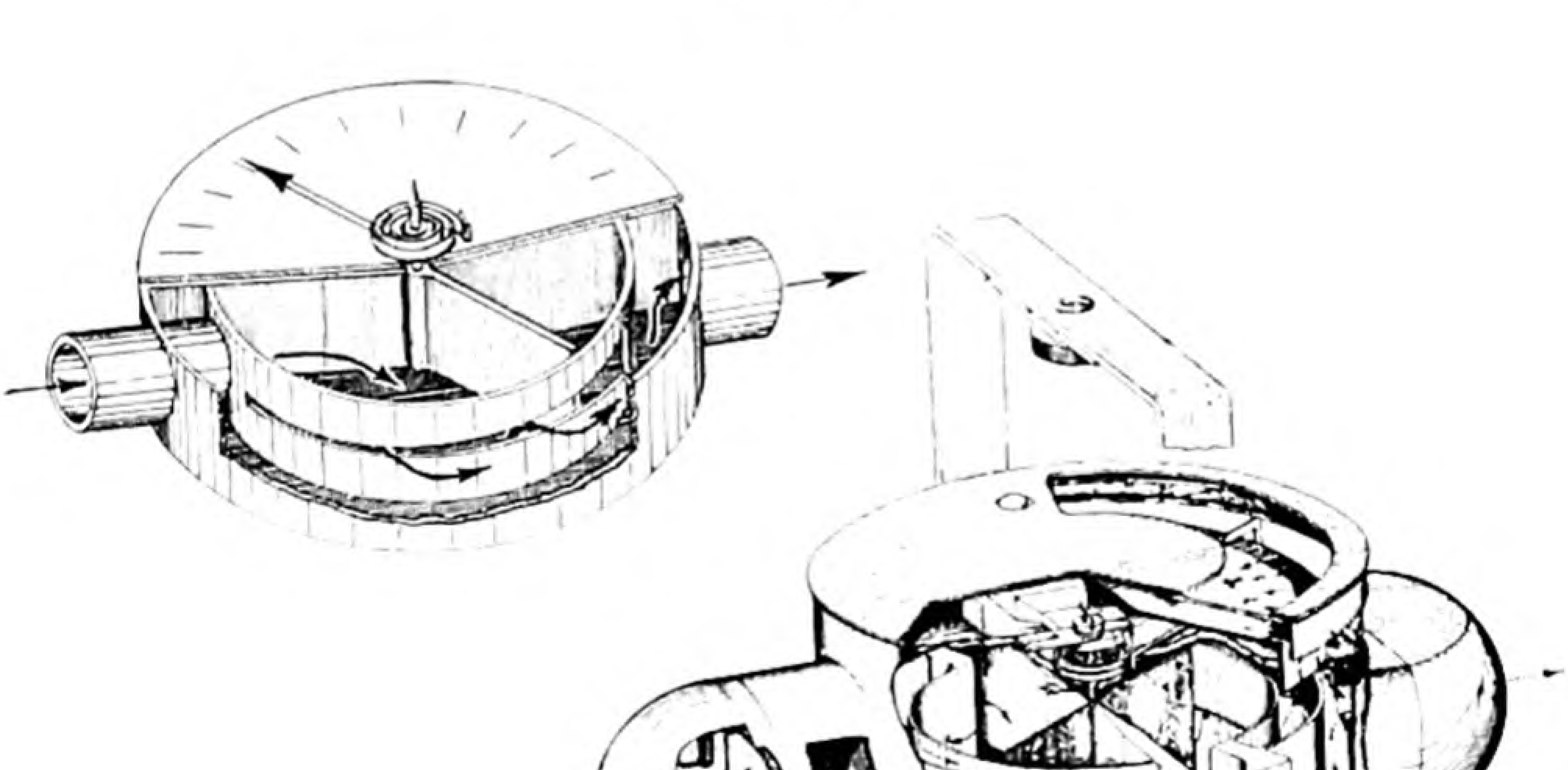


FIG. 377

velocity meter. This latter instrument does not rely for its operation upon the air-flow taking place round the swinging plate but along the plate, as shown in Fig. 377. The air flows along the plate and leaves the inner chamber through a slit in the wall without forming eddies which may cause fluttering of the plate. The measuring chambers are fitted with mesh filter screens capable of retaining the coarse dust. The fine dust is trapped in the inlet port before entering the actual measuring chamber by repeated deviation of the air-stream and enlargement of the inlet section. The pointer is rigidly attached to the plate axle and the deflection is read on a linear scale. The instrument is calibrated and the actual air velocity taken from the calibration curve. The instrument is used in the same manner as the velometer, and readings are recorded at numerous points in the roadway section. This meter has also been designed with an external clockwork-rotated drum to take continuous velocity recordings.

The air velocity may be obtained by measuring the velocity pressure of the air-stream with a pitot tube and a sensitive manometer. This method is suitable only where high velocities of over 1,000 feet per minute have to be recorded. The pitot static tube combination consists essentially of two parts. One part is an open-ended tube or facing gauge, which faces the air current and transmits the total head to the manometer. The other part differs in various instruments, but consists of a tube with side holes to register the static pressure of the air. These two tubes may be quite separate or can be combined and arranged concentrically with each other. The velocity pressure can be directly recorded on the manometer by coupling both sides of the manometer to the two connections of the pitot tube.

Section 2. Pressure Measurements

In the measurement of the ventilation pressure, the static pressure, velocity pressure and the combination of the two, namely total pressure, must be distinguished. The pitot static tube previously described can accurately measure air speeds above 1,000 feet per minute.

The velocity pressure may be defined as the energy due to the movement of the air, and is dependent on the density and velocity of the air and the acceleration due to gravity. The velocity pressure can be represented by the general equation:

$$P_v = \frac{W \times v^2}{2 \times g},$$

where v is the velocity of flow and W is the weight per unit volume. When taking air measurements, the temperature, humidity and barometric pressure of the atmosphere must also be observed. These variables may be combined to give the air density.

The static pressure cannot be defined in simple terms, since the value of the static pressure is different when considering exhausting or forcing ventilation systems. The diagrams in Fig. 378 illustrate

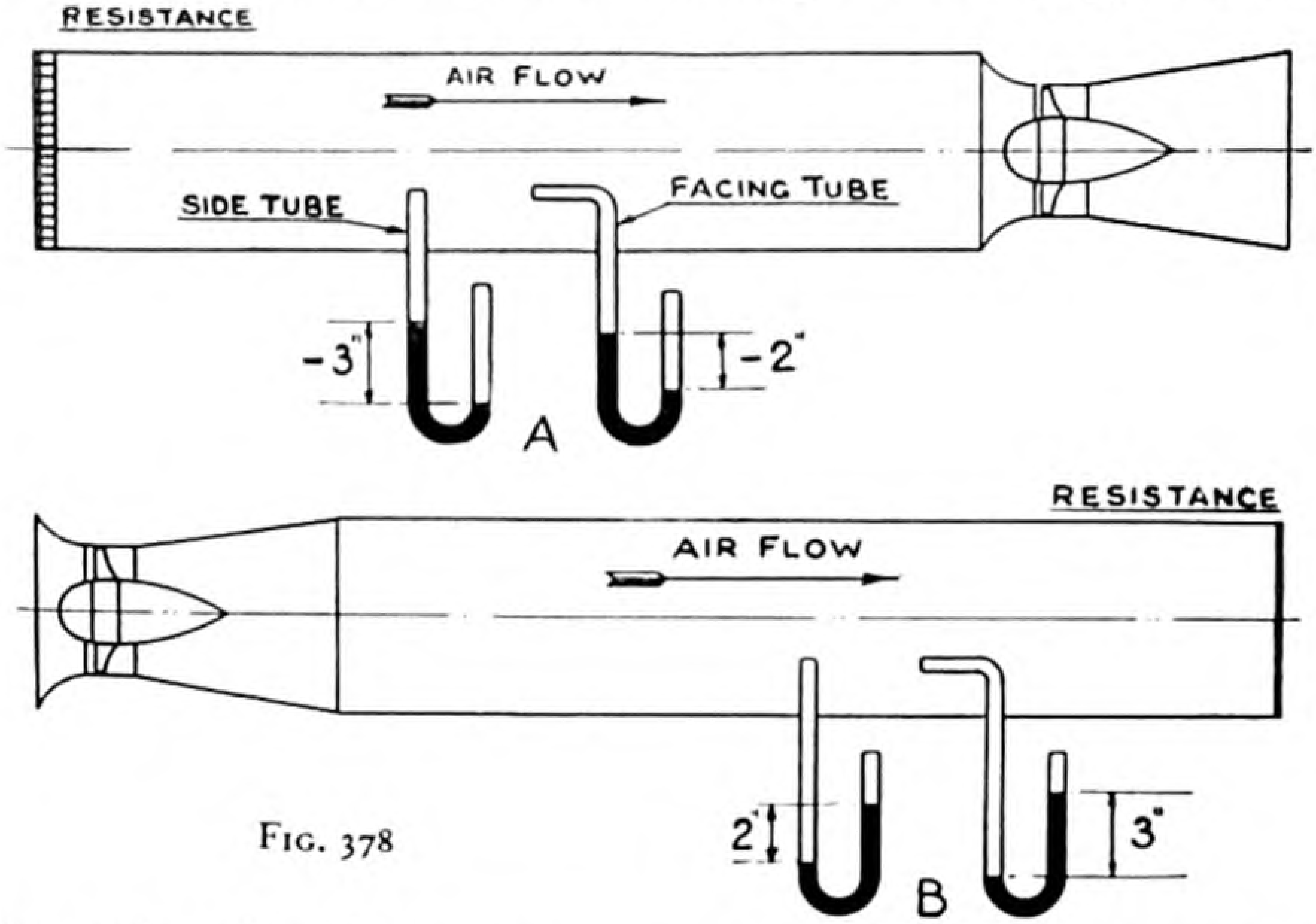


Fig. 378

the static and total pressure measurements in a fan duct—A when exhausting and B when forcing.

Static pressure may be defined as the pressure required to overcome resistance, but this is true only for a forcing-fan installation and is erroneous when used on a suction or exhaust system. Fan manufacturers overcome this difficulty in an exhausting system by referring to the pressure measured on a facing gauge as 'useful water gauge' or the head necessary to overcome the mine resistance. Thus, the overall total head which the fan has to set up is the useful water gauge plus the velocity head necessary to discharge the air into the atmosphere.

When the mine resistance is on the suction side of the fan there is a loss of pressure in the air due to this resistance, and the static

pressure, as measured by the side gauge of a pitot tube, is required to generate the velocity of the air and to give the head necessary to overcome the frictional resistance. The velocity head is opposite in sign to the static head generating the velocity.

The pressure as measured by the facing gauge on the suction side is therefore the component of the static pressure to overcome resistance. Thus, the total pressure P_t remains the algebraic sum of the static and velocity pressures, i.e.

$$-P_t = -p_s + p_v;$$

therefore,

$$p_v = p_s - P_t:$$

which shows that the static pressure P_s is larger than the total pressure P_t.

It is immaterial to the performance of the fan whether the resistance is on the suction or exhaust side—that is, the fan delivering a fixed volume of air at constant temperature and running at a constant speed will set up the same overall total water gauge no matter what the conditions of working.

The overall total head in the case of the fan in Fig. 378 (A)* when *exhausting* is thus:

$$\begin{aligned} P_t &= (p_s^1 + p_v^1) - (p_s + p_v) \\ &= (0 + 1) - (3 + 1) \\ &= 3 \text{ inches (side tube reading)}, \end{aligned}$$

$$\begin{aligned} \text{where } p_s^1 &= \text{side tube reading on discharge side;} \\ p_v^1 &= \text{velocity pressure on discharge side;} \\ p_s &= \text{side tube reading on suction side;} \\ p_v &= \text{velocity pressure on suction side;} \\ P_t &= \text{overall total pressure.} \end{aligned}$$

The velocity heads cancel out since there is no change in the kinetic energy across the fan. Also, when the fan is *forcing* under similar conditions:

$$\begin{aligned} P_t &= (p_s^1 + p_v^1) - (p_s + p_v) \\ &= (2 + 1) - (0 + 0) \\ &= 3 \text{ inches}, \end{aligned}$$

which is the algebraic sum of the total heads on each side of the fan as before.

* *Mine Ventilation*, D. MacFarlane. Paper read to the North of England Inst. Min. and Mech. Engrs., 16/12/44, published by Davidson & Co. Ltd., Belfast.

When measuring the total pressure difference along an airway, the velocity pressure must be considered where the velocities are high. In general, the determination of the static pressure is sufficient.

The static pressure difference between different points along a mine airway can be measured indirectly or directly.

In the indirect method, aneroid barometers are used to measure the barometric pressure at the different points. The difference of two readings can be converted from readings in inches of mercury to give the static pressure in inches water gauge after taking into account changes in level, temperature and humidity. The method is easily applied to survey the roadways comprising the ventilation system, but care must be taken with the instruments, which are subject to errors due to 'creep' and 'lag' in recording the barometric pressure.

The direct method utilises sensitive manometers and a connection between any two points with a rubber pressure hose. The tube-and-gauge method is considered to be more accurate than the barometric method, since the instruments are not affected to the same extent by instrumental difficulties and sudden changes in pressure caused by the opening of doors or winding in the main shaft. The hose may be in lengths up to about 300 yards, but 100 yards is probably most suitable. One end of the hose is fitted with a static head, while the other is coupled to the manometer. In horizontal roads the pressure difference readings can be made relatively quickly, waiting sufficiently long to ensure that the temperature of the air in the hose is the same as in the roadway section under consideration. The method has the disadvantage that it requires heavier and more complicated equipment, and takes more time than the indirect method. In horizon-mining practice this disadvantage is more noticeable, since the intake and return levels are widely separated. In this case the barometric method is generally preferred.

There are two things to be borne in mind when taking pressure measurements:

(a) *The influence of barometric changes on pressure differences measured along airways.* The pressure difference measured along an airway is independent of the total ventilation pressure. The measured difference changes, however, if there is an alteration in barometric pressure during the period of measurement. With a rising barometer the pressures recorded will be increased, and this is especially important in the indirect method of measurement, when

the barometer readings at the initial or base station, to which further readings are referred, must be continuously recorded. Where the tube and gauge method is employed, the static pressure is measured simultaneously at both ends of the tube and barometric changes in pressure do not affect the individual pressure differences.

(*b*) *The influence of changes in level on static pressure measurements.* In horizontal roadways, the measured pressure difference need not be adjusted for any change in the level between the two points considered. Where the roadways are inclined or in staple or main shafts, the difference in level between the two stations must be considered. When the pressure survey is proceeding in the direction of the air current and the downstream point is at a higher level, the pressure recorded at the downstream station will be too low by the weight of the air column between the two points. The weight of the air column must therefore be added to the measured value at the downstream point. The reverse is the case when the downstream station is at a lower level, and the recorded pressure at the downstream station must be correspondingly reduced.

The pressure due to the air column can be computed if the average air density is known between the two points.

The weight of the air column is computed and the necessary correction applied to the recorded pressure difference according to the relation—

$$P = \frac{wH}{5 \cdot 2},$$

where P is the pressure in inches W.G., w is the weight of a cubic foot of air and H is the difference in level in feet between the two points.

Section 3. Temperature and Humidity Measurements

The condition of the air must be known in order to apply the correction to the static pressure values as previously described, to obtain the data from which to calculate the natural ventilation pressure and, if necessary, to compute actual velocities of flow from velocity pressure measurements.

The degree of saturation of the air can be obtained using the wet and dry bulb whirling hygrometer. The wet and dry bulb temperatures are used to determine the percentage saturation or relative humidity from hygrometric tables. The less the degree of saturation, the greater is the difference between dry and

wet bulb readings, the wet bulb giving the lower reading. As it is not possible to whirl the thermometer always at the same velocity, results shown by the whirling hygrometer may be erratic.

The weight of a cubic foot of dry air can be calculated from the equation:

$$w = \frac{1{\cdot}325(b - 0{\cdot}378e)}{459 + t},$$

where w is the weight in pounds of one cubic foot of air at the barometric pressure b in inches of mercury and temperature t in ° F. If the relative humidity is r and the vapour pressure corresponding to t is f, then $rf = e$. The formula shows that the density of the air increases with rising barometric pressure and decreases with rising temperature.

PART III

VENTILATION THEORY AND COMPUTATIONS

Section 1. General Introduction. The Calculation of Static Pressure Differences

The quantity of air and the static difference of pressure of airways (shafts, roads, coal faces) can be determined by measuring. As will be shown, once the values of the coefficients appearing in the formulæ are known, it is possible to compute in advance the influence on the ventilation conditions of a possible change in length or cross-section of airways and of future stages of development of a whole mine, or parts of it.

Since it can be considered that the air-flow in underground roadways comes within the turbulent flow condition, the general equation connecting pressure with the characteristics of the airway and the air-flow can be written as

$$P = \frac{k\,c\,l\,v^2}{A} = \frac{k\,s\,v^2}{A},$$

which is derived from a hydraulic formula for turbulent flow and in which P = difference in total pressure between two points in the airway, lb. per square foot, k = coefficient of friction obtained experimentally, s = rubbing surface, square feet, c = perimeter,

feet, $l =$ length, feet, $v =$ velocity of air-stream, feet per minute, and A is the average cross-section, square feet.

The equation may be modified to—

$$P = \frac{k\,c\,l\,Q^2}{A^3},$$

where P is the pressure loss in lb. per square foot due to friction losses and Q is the quantity flowing in cubic feet per minute. The equation shows that the pressure drop is dependent upon the length, perimeter and area of the airway. The pressure drop will be greater the longer the roadway and the greater the perimeter, and will become smaller as the area is increased. The pressure drop is proportional to the square of the velocity, so that if the velocity is doubled, the pressure drop is quadrupled. The pressure drop is also proportional to the coefficient of friction k. All other values in the equation can be determined readily except the coefficient k.

Modern research on air-flow has shown that the flow conditions are identical in airways or ducts which are geometrically and dynamically similar. Geometric similarity exists if the surfaces of the two ducts being compared are also similar. Thus comparisons can be made between different sections of airway with similar linings. Dynamic similarity exists if the Reynolds number is the same in both cases. The Reynolds criterion introduces a dimensionless constant in terms of diameter, velocity, density and viscosity, and can be computed as follows:

$$R_e = \frac{v\,d}{\nu},$$

where $v =$ mean velocity in cms. per second;

$d =$ diameter of duct in cms.

$$\nu = \text{kinematic viscosity} = \frac{\text{viscosity of fluid}}{\text{density of fluid}}.$$

or, introducing the aerodynamic diameter d_a for ducts which are not circular

$$R_e = \frac{vd_a}{\nu}$$

$$\text{where } d_a = \frac{4A}{c}$$

Section 2. The Calculation of Airway Resistances

The calculation of the equivalent resistance of the various ventilating circuits has been found to be sufficient for practical requirements. The equivalent can be derived from the formula previously given, provided that the influence of change in the Reynolds number is not considered; thus—

$$P = \frac{k\,c\,l\,Q^2}{A^3}$$

$$= R\,Q^2 \text{ or } R = \frac{P}{Q^2},$$

where P is the pressure in lb. per square foot, Q is the quantity passing in thousands of cubic feet per second and R is the resistance in Atkinsons.

The Atkinson is defined as the resistance which absorbs a pressure of 1 lb. per square foot when a volume of 1 kilocusec of dry air at 60° F. and 30 inches barometer is flowing. The kilocusec is a quantity of 1,000 cubic feet per second.

The value of resistance can be considered in terms of the electrical analogy to Ohm's Law, but is distinguished from it by the approximate quadratic dependency of Q valid for turbulent flow.

When the pressures and quantity measurements have been made in an underground airway, the value of the airway resistance can be easily calculated. Similarly, any change in the pressure can be related to the quantity of air flowing and vice versa. The use of calculations of this nature for future development is obviously important.

Specific details of the various roadway resistances is of great importance when planning new mine developments. Experience over several years can indicate the influence of various methods of support roadway lining and cross-sectional area on the value of the roadway resistance. These resistances can be based on a standard length of, say, 1,000 feet and the resistances compared. The following table extracted from the N.C.B. Bulletin No. MP (57)14 shows the resistance in Atkinsons per 1,000 feet of roadway for various cross-sectional areas.

Resistance in Atkinsons per 1,000 ft. of roadway

Cross-sectional area of airway	Supported by steel arches				Rectangular airways		
	Bricked all round	*Bricked to spring*	*Timber or steel lagging (main airways)*	*Timber or steel lagging (gate roads)*	*Concrete lined*	*Girders on brick walls*	*Wooden bars or steel girders on wooden legs*
sq. ft.	k. = ·0030	k. = ·0040	k. = ·0055	k. = ·0065	k. = ·0020	k. = ·0050	k. = ·0100
30	8·29	11·05	15·21	17·98	5·96	14·90	29·80
40	4·06	5·41	7·44	8·79	2·91	7·28	14·56
50	2·32	3·09	4·25	5·02	1·66	4·16	8·32
60	1·47	1·96	2·70	3·19	1·06	2·64	5·28
70	1·00	1·34	1·84	2·17	0·72	1·80	3·60
80	0·72	0·95	1·31	1·55	0·51	1·29	2·57
90	0·54	0·71	0·98	1·16	0·38	0·96	1·92
100	0·41	0·55	0·75	0·89	0·30	0·74	1·48
110	0·32	0·43	0·59	0·70	0·23	0·58	1·16
120	0·26	0·34	0·47	0·56	0·186	0·47	0·93
130	0·21	0·28	0·39	0·46	0·152	0·38	0·76
140	0·18	0·23	0·32	0·38	0·126	0·32	0·63
150	0·15	0·20	0·27	0·32	0·106	0·27	0·53
160	0·12	0·17	0·23	0·27	0·090	0·23	0·45
180	0·092	0·12	0·17	0·20	0·066	0·17	0·33
200	0·072	0·097	0·133	0·157	0·052	0·13	0·26
220	0·057	0·076	0·104	0·123	0·041	0·102	0·204
240	0·046	0·061	0·084	0·099	0·033	0·082	0·164
260	0·037	0·050	0·069	0·081	0·027	0·067	0·134
280	0·031	0·041	0·057	0·067	0·022	0·056	0·111
300	0·026	0·034	0·047	0·056	0·019	0·047	0·093

Each of the separate ventilation circuits comprising the mine ventilation system consist of airways with different resistances. The combined resistance of the several sections is frequently required, the total resistance being composed of numerous single and compound resistances in series and in parallel. The computation of the total resistance of a number of airways in series is given by the expression

$$R = R_1 + R_2 + R_3 \; . \quad . \quad . \quad + R_n$$

If the airways are connected in parallel, then

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} \; \cdot \; + \frac{1}{\sqrt{R_n}}$$

Since the quantity flowing in a split is inversely proportional to the square root of the resistance of the split, then

$$\frac{Q_1}{Q_2} = \sqrt{\frac{R_2}{R_1}}$$

Thus, from these expressions the total resistance of the series can be computed. The following examples, taken from practice, illustrate the use of the expressions and indicate the possibility of calculating the changes in ventilation conditions after a reorganisation of the system.

Example 1. A third shaft is to be sunk at a mine having single downcast and upcast shafts. The change in the ventilation conditions consequent upon the sinking of the third shaft, which will act as a downcast, can be computed as follows: The total water gauge is 8 inches, of which 2 inches is being absorbed by the present downcast, the pressure across the shaft bottom between the upcast and downcast shafts, i.e. across the mine excluding the shafts, is 4 inches, and the pressure absorbed in the upcast shaft is 2 inches. The total quantity of air passing through the mine is 240,000 cubic feet per minute.

Downcast shaft resistance $= R_1 = 0{\cdot}65$ Atk.
Resistance of all the other airways $= R_2 = 1{\cdot}95$ Atk.

If the second downcast shaft is sunk near the present downcast, both shafts can be considered to be in parallel. The combined resistance or the two downcast shafts is given by—

$$\frac{1}{\sqrt{R_3}} = \frac{1}{\sqrt{0{\cdot}65}} + \frac{1}{\sqrt{0{\cdot}65}} \text{ and } R_3 = 0{\cdot}16 \text{ Atk.}$$

The mine resistance including the upcast will be—

$$R = R_2 + R_3 = 1{\cdot}95 + 0{\cdot}16 = 2{\cdot}11 \text{ Atk.}$$

The quantity of air passed by the fan will increase due to the reduction in the total resistance and, assuming the same total water gauge, the new quantity is given by—

$$P = RQ^2$$

and $Q = 266{,}000$ cubic feet per minute,

thus showing an increase from 240,000 cubic feet per minute of

26,000. If, however, the downcast shaft absorbs 4 inches W.G. or half the total available water gauge, then

$$\text{Downcast shaft resistance} = R_1 = 1\cdot30 \text{ Atk.}$$
$$\text{Resistance of all the other airways} = R_2 = 1\cdot30 \text{ Atk.}$$

Considering the resistance of the new shaft to be 0·65 Atkinson, the combined resistance of the shafts in parallel will be—

$$\frac{1}{\sqrt{R_3}} = \frac{1}{\sqrt{0\cdot65}} + \frac{1}{\sqrt{1\cdot30}} \text{ and } R_3 = 0\cdot22 \text{ Atk.}$$

The total mine resistance will then equal—

$$R = R_2 + R_3 = 1\cdot30 + 0\cdot22 = 1\cdot52 \text{ Atk.}$$

With reduced resistance at the total available water gauge of 8 inches, the new quantity flow in the mine is—

$$Q = \sqrt{\frac{8 \times 5\cdot2}{1\cdot52}} \text{ kilocusecs} = 314{,}000 \text{ cubic feet per minute.}$$

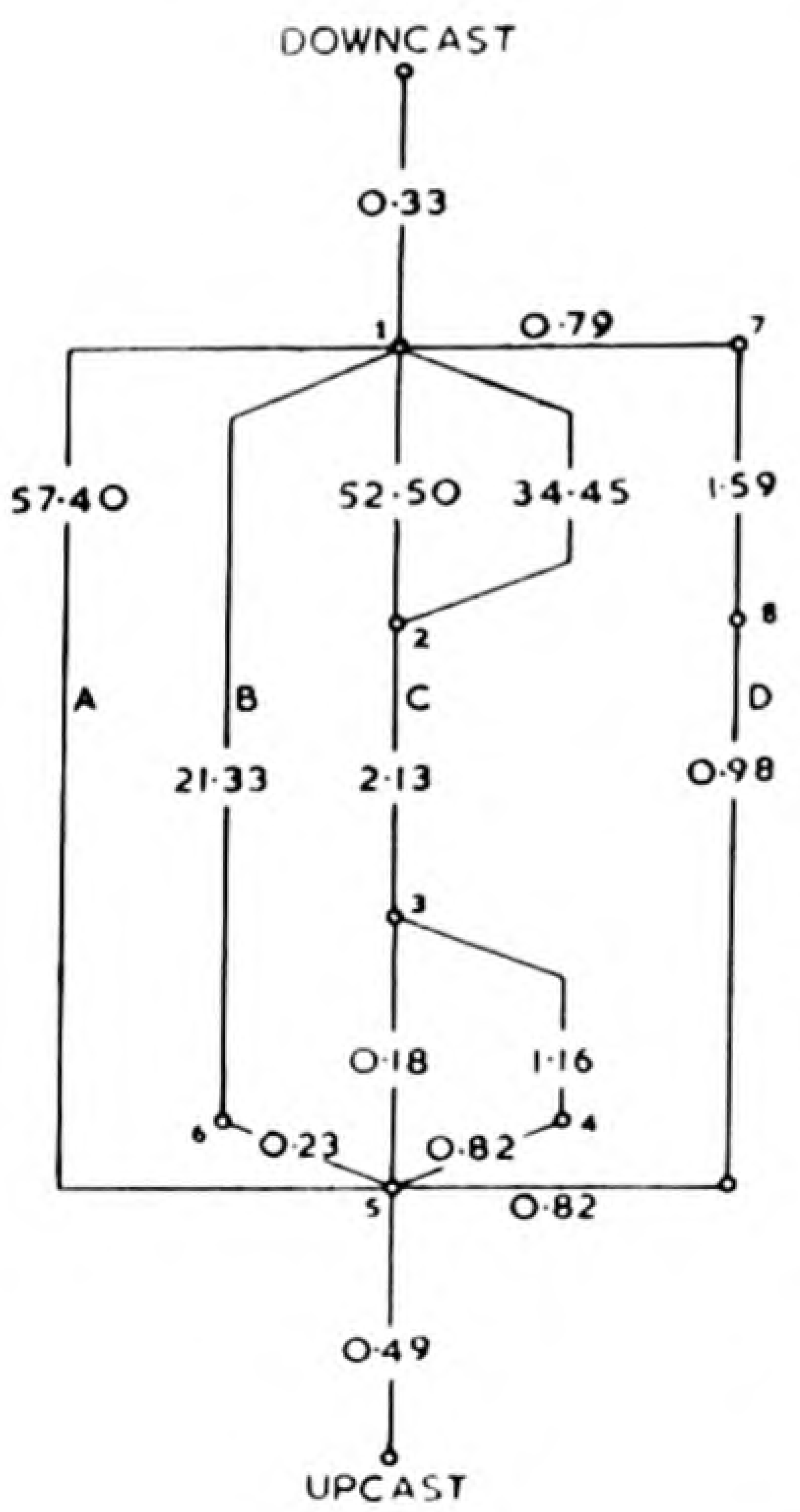


FIG. 379

Example 2. Further examples may be illustrated with reference to the 'flow-net' represented in Fig. 379. In the diagram, the downcast and upcast shafts are shown, together with airways A, B, C and D, comprising four parallel circuits. There are also two short airways 1, 2, and 3, 4, 5 shown in parallel with the flow through circuit C. The resistance of each section has been calculated on the basis of the measured values of the average quantity of air passing and the pressure drop across each circuit, and entered on the diagram.

In order to calculate the total resistance of the mine, it is necessary to determine the individual resistances of the four parallel circuits. The calculation is simple for airways A, B and D, which have

several resistances in series. The resistance of circuit C includes two additional splits in parallel, so that—

$$\frac{1}{\sqrt{R_{1\text{-}2}}} = \frac{1}{\sqrt{52{\cdot}50}} + \frac{1}{\sqrt{34{\cdot}45}}$$

$$\text{and } R_{1\text{-}2} = 10{\cdot}53 \text{ Atk.}$$

Similarly, the resistance of $R_{3\text{-}5}$ is given by—

$$\frac{1}{\sqrt{R_{3\text{-}5}}} = \frac{1}{\sqrt{0{\cdot}18}} + \frac{1}{\sqrt{1{\cdot}16 + 0{\cdot}82}}$$

$$\text{and } R_{3\text{-}5} = 0{\cdot}11 \text{ Atk.}$$

The combined resistances having been computed, the series combination in circuit C can be calculated.

$$\begin{aligned} R_{1\text{-}2\text{-}3\text{-}5} &= 10{\cdot}53 + 2{\cdot}13 + 0{\cdot}11 \\ &= 12{\cdot}77 \text{ Atk.} \end{aligned}$$

The four circuits in parallel can be combined.

$$\begin{aligned} \frac{1}{\sqrt{R_u}} &= \frac{1}{\sqrt{57{\cdot}40}} + \frac{1}{\sqrt{21{\cdot}33 + 0{\cdot}23}} + \frac{1}{\sqrt{12{\cdot}77}} + \frac{1}{\sqrt{0{\cdot}79 + 1{\cdot}59 + 0{\cdot}98 + 0{\cdot}82}} \\ &= 0{\cdot}13 + 0{\cdot}22 + 0{\cdot}28 + 0{\cdot}49 \\ &= 1{\cdot}12 \end{aligned}$$

and $R_u = 0{\cdot}80$ Atk.

Since the shaft resistances are also known, the total mine resistance can be computed from the relationship—

$$\begin{aligned} R_M &= R_{0\text{-}1} + R_u + R_{5\text{-}6} \\ &= 0{\cdot}33 + 0{\cdot}80 + 0{\cdot}49 \\ &= 1{\cdot}62 \text{ Atk.} \end{aligned}$$

The influence of changes in the ventilation system can also be illustrated.

Example 3. The resistance of airway A may be reduced from 57·40 Atkinsons to, say, 16·40 Atkinsons by enlarging the sectional

area and modifying the supports. The new resistance for the four parallel airways is therefore—

$$\frac{1}{\sqrt{R_u}} = \frac{1}{\sqrt{16\cdot40}} + \frac{1}{\sqrt{21\cdot33 + 0\cdot23}} + \frac{1}{\sqrt{12\cdot77}} + \frac{1}{\sqrt{0\cdot79 + 1\cdot59 + 0\cdot98 + 0\cdot82}}$$

$$= 0\cdot25 + 0\cdot22 + 0\cdot28 + 0\cdot49$$

$$= 1\cdot24$$

and $R_u = 0\cdot65$ Atk.

The new mine resistance is thus—

$$R_M = 0\cdot33 + 0\cdot65 + 0\cdot49$$
$$= 1\cdot47 \text{ Atk.}$$

The reduction of the total mine resistance from 1·62 Atkinsons to 1·47 Atkinsons will allow the fan water gauge to be reduced or, if the same water gauge is maintained, the quantity of air passing in the mine will be increased. With a water gauge of 6 inches and a mine resistance of 1·62 Atkinsons, the quantity passing is 263,000 cubic feet per minute, while with a resistance of 1·47 Atkinsons the quantity is increased to 276,000 cubic feet per minute. This increased quantity of air will to some extent benefit the airway A, but if this increase in the quantity is not required, then the total water gauge can be reduced. The reduced resistance of airway A implies an increase in the quantity of air passing in that airway, while the share taken by B, C and D will be slightly reduced.

Example 4. The airway A is to be omitted from the system. The combined resistance of the remaining airways in parallel will then be 1·02 Atkinsons. Adding the shaft resistances to this value, the total mine resistance is 1·84 Atkinsons. In order to supply the mine with the same total quantity of 263,000 cubic feet per minute as in the original condition, the available water gauge has to be increased to—

$$P = \frac{1\cdot84}{5\cdot2} \times \frac{(263{,}000)^2}{(60{,}000)} = 6\cdot8 \text{ inches W.G.}$$

It is probable, however, that the total quantity of air to be passed by the fan can be reduced by the quantity which originally entered

split A, amounting to 43,000 cubic feet per minute. The total quantity of air required is, therefore, 220,000 cubic feet per minute, which with a mine resistance of 1·84 Atkinsons will require 4·7 inches W.G. at the fan to circulate this quantity round the mine.

Example 5. If a fifth parallel airway E of resistance 20·03 Atkinsons is added to the system, the combined resistance of the splits will be 0·56 Atkinson and the whole mine 1·38 Atkinsons. This is a condition similar to that obtained in the calculation for Example 3, in which the resistance of airway A was reduced from 57·40 to 16·40 Atkinsons. The difference in condition is that airway E requires an additional quantity of, say, 50,000 cubic feet per minute. The total quantity of air required is therefore to be increased to 313,000 cubic feet per minute, which requires a water gauge of 7·2 inches when the mine resistance is 1·38 Atkinsons. The change in the water gauge will result in an alteration in the splitting of the air in the other parallel airways. The usual method employed when one airway is restricting the quantity of air flowing in another is to install a regulator in the split in which the quantity flowing has to be reduced.

Electrical air-flow models have recently been introduced to predict the results of any alterations in the air-flow system, without making complicated and time-consuming calculations. The aerodynamic problem has an equation of the second degree (Atkinson's law $P = RQ^2$) and an equivalent to that was found in the relation between voltage and current in tungsten incandescent lamps between 25 per cent. and 100 per cent. of their normal range. It is necessary to test each type of lamp for this relation. Resistances with the same characteristics have lately been introduced, which allow for careful adjustment to the proper value. Once a model has been constructed, the quantity of air flowing at any point can be measured by a milliamperemeter and the pressure difference between two points by a voltmeter.

Section 3. *The Equivalent Orifice Theory*

Since the flow through an orifice follows the same law as flow through a mine, it has been suggested that the mine resistance can be represented by the size of an orifice in a thin plate which will pass the same quantity with the same pressure loss. This orifice for a fan is called the 'orifice of passage' and for a mine the 'equivalent orifice,'

and can be obtained from the formula—

$$q = C_d A\sqrt{2gh} \text{ or } A = \frac{q}{C_d\sqrt{2gh}}$$

in which the coefficient of discharge, C_d, is taken as 0·65. The pressure head, h, is the height of air column causing a flow of q cubic feet of air per second. This expression assumes constant air density and, if an average air density of w lb. per cubic foot is taken, the expression may be written in the more acceptable form, including the velocity in feet per second and the water gauge in inches:

$$v = C_v\sqrt{2gh} = C_v\sqrt{2g\frac{\text{W.G.} \times 5\cdot 2}{w}}$$

Assuming the coefficient of velocity, C_v, as unity and that the average air density is 0·078 lb. per cubic foot—

$$v = \sqrt{2 \times 32\cdot 2 \times \frac{\text{W.G.} \times 5\cdot 2}{0\cdot 078}} \text{ feet per second.}$$

$$= 65\cdot 5\ \sqrt{\text{W.G.}} \text{ feet per second.}$$

Introducing the quantity Q in cubic feet per minute, then

$$q = C_d vA \text{ cubic feet per second.}$$

$$\therefore A = \frac{Q}{2555\sqrt{\text{W.G.}}} = \frac{0\cdot 00039\ Q}{\sqrt{\text{W.G.}}}$$

taking $C_d = 0\cdot 65$.

Expressing Q in thousands of cubic feet per minute—

$$A = \frac{0\cdot 39\ Q}{\sqrt{\text{W.G.}}} \text{ square feet.}$$

This formula is not quite accurate, since the water gauge depends on the Reynolds number and thus on the air velocity, but it is sufficiently correct for all practical purposes. The equivalent orifice will also be affected by the influence of natural ventilation, and when the expression is applied, the exact area of orifice can be obtained only by including the natural ventilation effect.

The expression shows that the equivalent orifice and the corresponding pressure difference will remain constant for all roadways in which the cross-sectional area, linings and distribution remain the same. This fact is of particular assistance in the supervision of the ventilation, especially in controlling the working of the fan. The

equivalent orifice at British mines and at the Ruhr collieries varies from 20 to 65 square feet.

It should be noted that in determining the equivalent orifice of a mine the quantity of air measured in the fan drift may include considerable leakage and the greater the apparent total quantity passing, the larger is the equivalent orifice, thus giving a false impression of the ventilation efficiency.

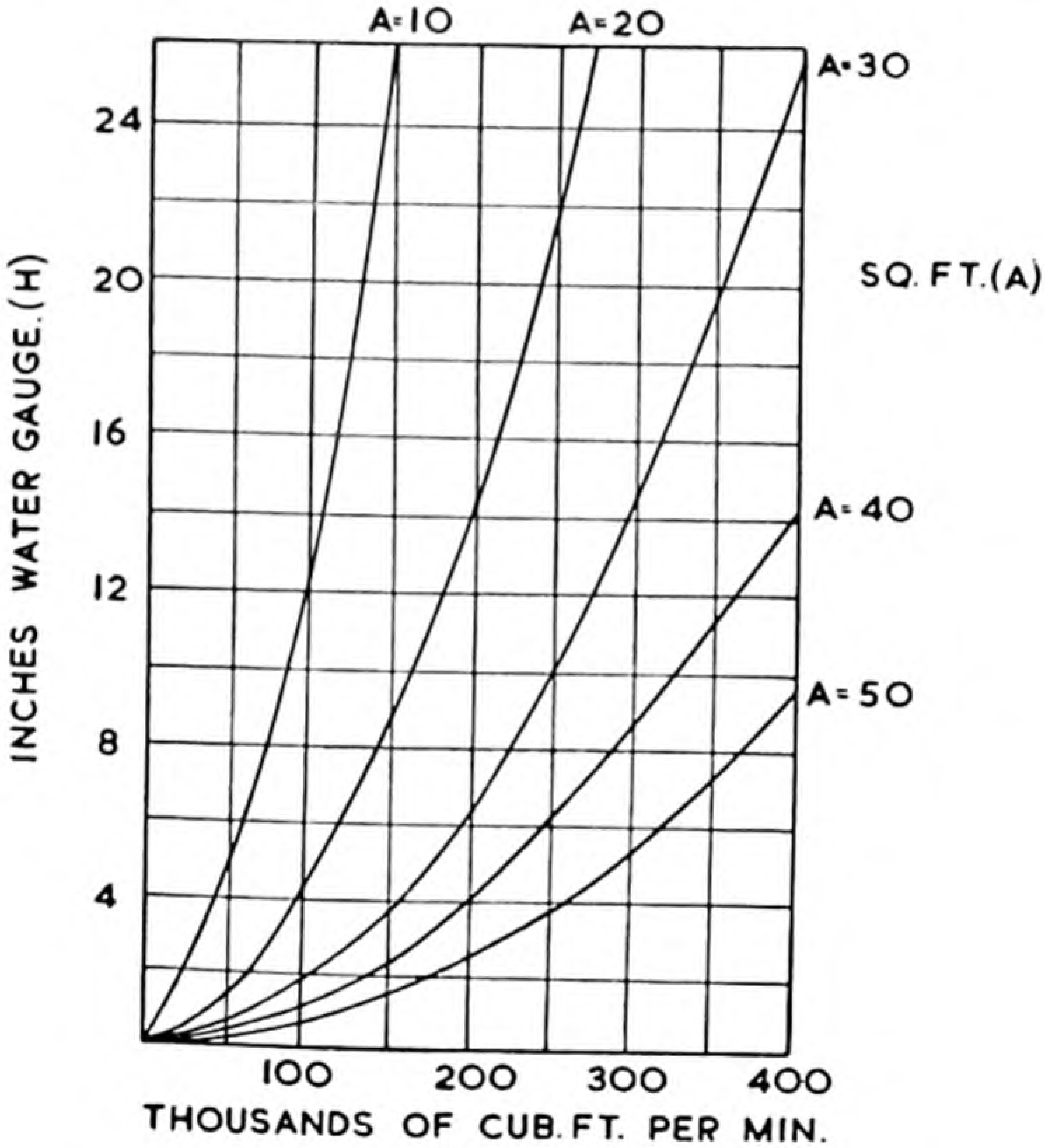


FIG. 380

The influence of the equivalent orifice on the quantity and pressure of air flowing, the fan horse-power and the mechanical efficiency of the fan are considered in the following sections:

(*a*) *The influence of the equivalent orifice on the quantity and pressure.* It can be seen from the expression for the equivalent orifice (A), in which the quantity (Q) is included, that the water gauge (h) is a function of Q. In Fig. 380 varying values for the equivalent orifice have been used to determine Q and h, resulting in a series of parabolic curves. Thus, each value of equivalent orifice has a definite

'family' curve for which that area of orifice is valid. The curve for $A = 20$ shows that in the case of a quantity of 100,000 cubic feet per minute, a pressure difference of 4·2 inches W.G. is required, and a water gauge of 14 inches and 32·6 inches if the quantity has to be doubled or trebled. Further comparison of the family curves for $A = 20$ and $A = 40$ show that where the ventilating pressures are the same, doubling the equivalent orifice will double the quantity passing.

On the other hand, doubling the quantity passing without increasing the equivalent orifice will require four times the ventilation pressure. The family curve for $A = 30$ illustrates that a quantity of 150,000 cubic feet per minute will require a water gauge of 3·6 inches and double this quantity will require 14·4 inches.

If the ventilation pressure is to be reduced, while the quantity passing remains constant, then an increase in the equivalent orifice will be required. A comparison of the family curves for $A = 20$ and $A = 40$ shows that when the equivalent orifice is doubled, the pressure required is reduced to 0·25. Thus, for a quantity of 150,000 cubic feet per minute with an orifice of $A = 20$, the water gauge required is 8·8 inches, while with an orifice of $A = 40$, a pressure of 2·2 inches W.G. is required.

(*b*) *The influence of the equivalent orifice on the fan horse-power.* The air horse-power at the fan can be calculated from the relationship—

$$\text{h.p.} = \frac{\text{Q.P.}}{33{,}000},$$

where Q is the quantity in cubic feet per minute and P is the ventilating pressure in lb. per square foot. Thus, the horse-power is proportional to the product of the quantity and pressure.

When the quantity remains the same and the equivalent orifice is doubled, the pressure is reduced to one-quarter, as h is inversely proportional to the square of the equivalent orifice. Thus the horse-power of the fan is dependent to a great extent on the equivalent orifice.

It is therefore essential, bearing in mind the initial capital cost for the fan installation and the power cost when the fan is installed, to make the equivalent orifice as large as possible. The mine resistance must be kept at a low value by constant roadway maintenance, efficient splitting of the air and choice of appropriate roadway sizes

for the passage of the quantity required. The air velocity must be kept to reasonable proportions and, in particular, the return airways should be adequate for the increased quantity which is passing.

Referring to the graph in Fig. 381, the horse-power decreases from 700 to 300, where a quantity of 200,000 cubic feet per minute is passing in a mine in which the equivalent orifice is increased from 20 to 30. An increase of A from 30 to 40 square feet

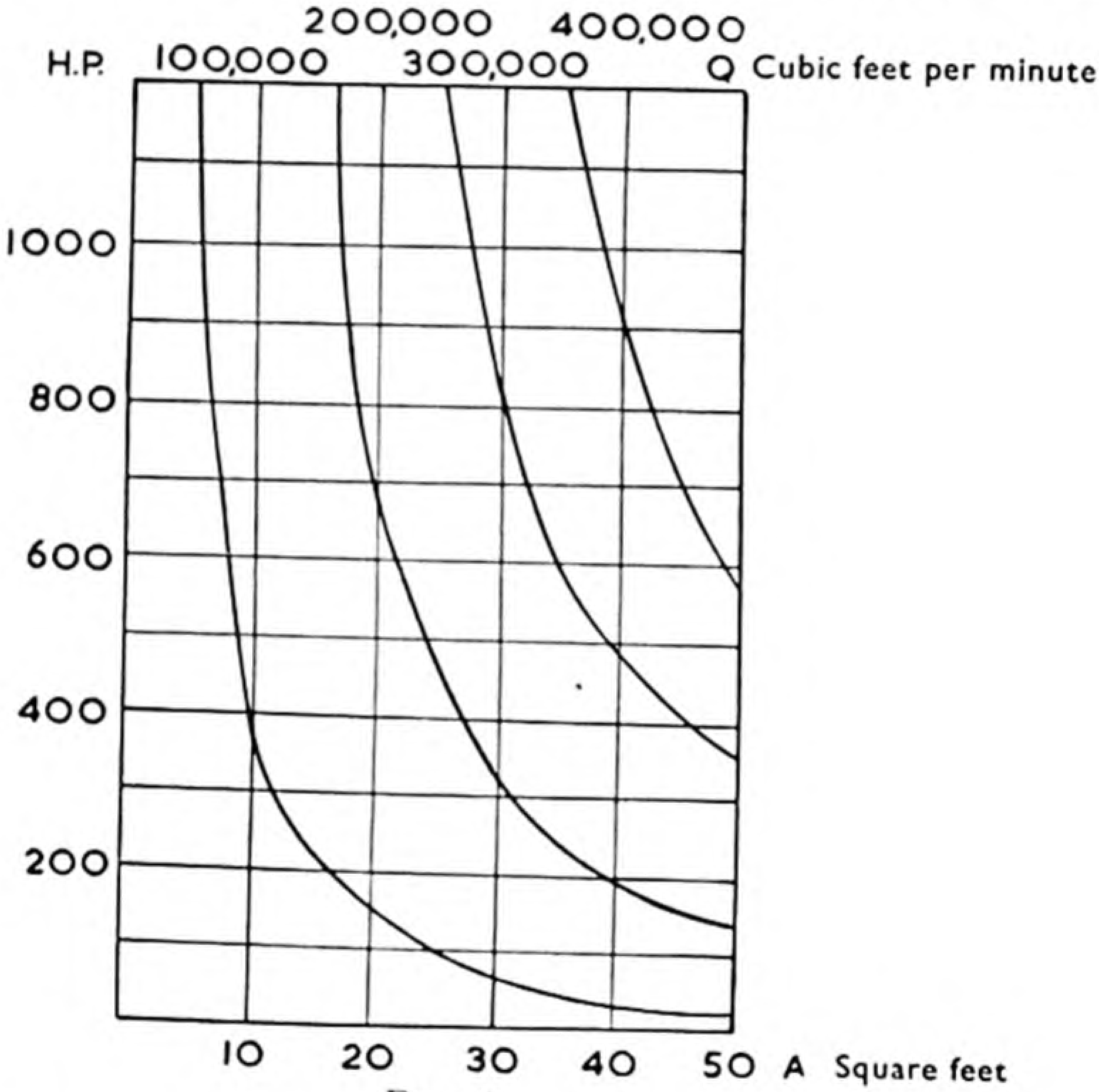


FIG. 381

will even effect a reduction from 820 to 440 h.p. when a quantity of 300,000 cubic feet per minute is circulating in the mine.

It should be pointed out, however, that the enlargement of the equivalent orifice beyond a certain optimum area will result in small improvements only, and a disproportionately large expense will produce very little gain. Thus, from the curve in Fig. 381, it is clear that with a quantity of 100,000 cubic feet per minute the enlargement of the equivalent orifice beyond 30 or 40 square feet will result in only a very small reduction in the pressure required. The cost of increasing the equivalent orifice should be compared with the in-

crease in ventilation efficiency. Thus, in each individual case, there is an upper limit for increasing the equivalent orifice, the limit being so much higher, the greater the quantity passing and the lower the costs for increasing and maintaining airways of large cross-sectional area.

(*c*) *The influence of the equivalent orifice on the mechanical efficiency of the fan.* The mechanical efficiency of a fan is measured by the ratio of the effective capacity of the fan in circulating the air in the

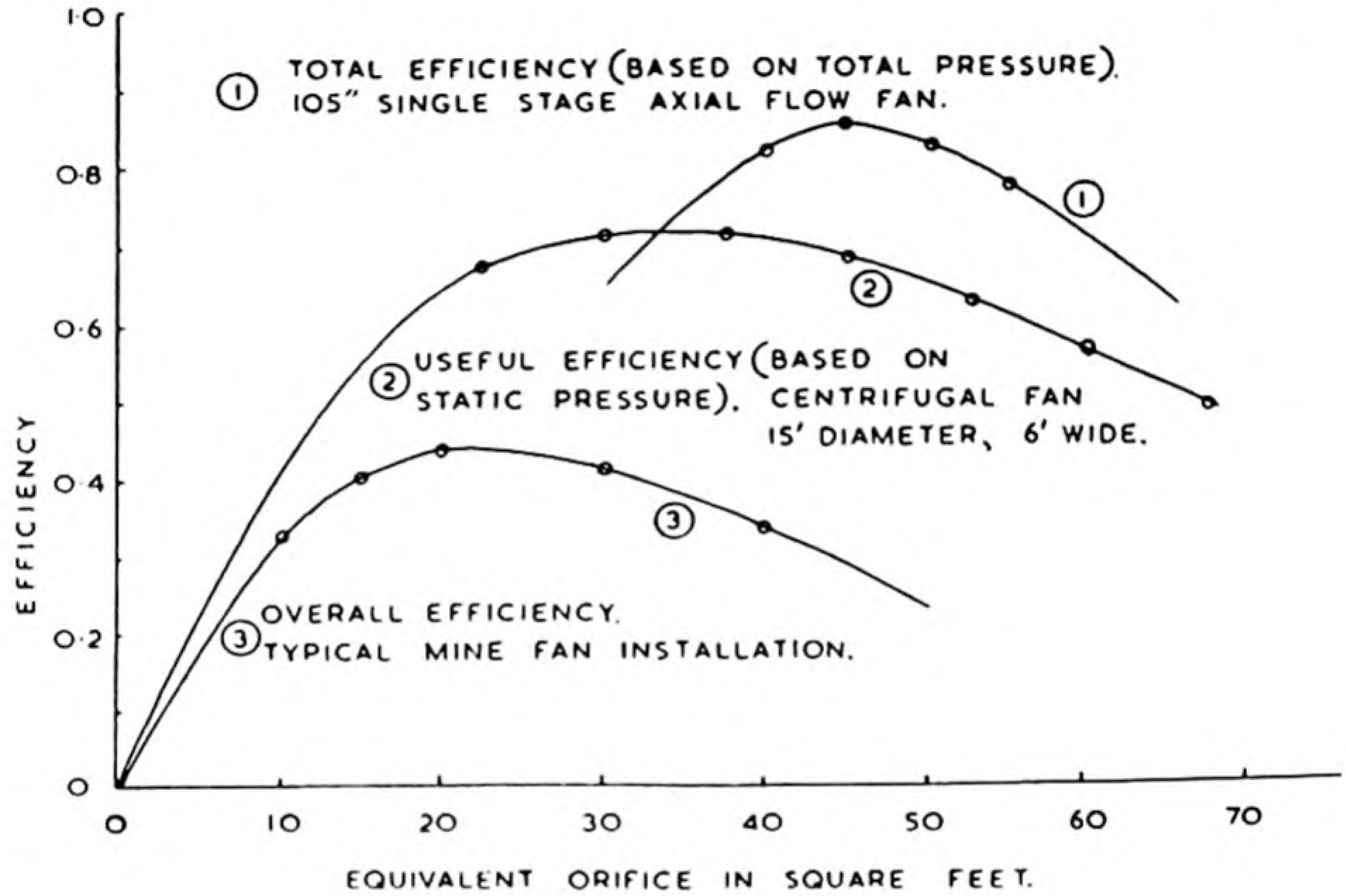


FIG. 382

mine to the horse-power supplied to the fan, or

$$\text{Mech. Efficiency} = \frac{\text{h.p. in the air}}{\text{h.p. of the fan}} \times 100$$

Thus, for example, a fan is delivering 100,000 cubic feet per minute against a total water gauge of 3 inches, and the b.h.p. of the driving unit is 78.

The mechanical efficiency of the fan is

$$\frac{3 \times 5{\cdot}2 \times 100{,}000}{33{,}000 \times 78} \times 100 = 60{\cdot}7 \text{ per cent.}$$

This value includes the efficiency of the fan and the efficiencies of the motor and drive. The fan efficiency is higher, and in many cases may be as high as 80 per cent. The actual fan efficiency is usually

supplied by the manufacturer from a works test. The effective efficiency of the fan is often lower than the example quoted. The efficiency depends to a great extent on the equivalent orifice, and many fans in use are not operating within the range of the equivalent orifice for which they were designed, so that poor efficiencies result.

The influence of equivalent orifice on fan efficiency is shown in Fig. 382. The curves illustrate that in the first instance the efficiency increases with increasing equivalent orifice, but that the fan has a 'peak' performance and the efficiency then decreases. It is obvious that a fan having a flat maximum characteristic will be more suitable than one which has a decided peak followed by a sudden decrease in efficiency. The fan having a flat characteristic curve is less likely to be affected by unavoidable changes in the equivalent orifice of the mine, while the other type will operate at a much lower efficiency.

The reason for the trend of the curves in Fig. 382 can be explained in the following way. If the suction side of the fan is closed and the equivalent orifice practically zero, the fan will create a high pressure but will not deliver any air, so that the fan efficiency will then be zero. When the suction area is increased, the fan will deliver an increasing quantity and the efficiency will gradually rise to a maximum value. If the suction opening is increased still further, the efficiency will be reduced. This will be clear if it is considered that the fan is operating with an infinitely large suction opening and delivering a large quantity of air at only a very small pressure, which finally approaches the zero value. The curve for mechanical efficiency therefore drops again with larger equivalent orifice areas, and will intersect the base of the graph at infinity.

PART IV

CHARACTERISTIC 'FAMILY' CURVES FOR A FAN

Family curves for a fan running at different speeds can be obtained in the same manner as described for the equivalent orifice, based on the variation in the values for the quantity passed by the fan and the pressure generated. This relationship can be made clear if it is understood that the fan will regulate the quantity delivered according to the resistance, or pressure difference, against which it is operating. Thus, if the pressure difference is changed and the quantity supplied at several pressures is measured, a number of points of the P–Q

diagram can be obtained to produce a characteristic family curve for the particular fan. Due to the relationship between the quantity flowing, pressure developed and speed of rotation of the fan, a particular curve can be obtained for each fan speed.

The majority of these curves are similar to those shown in Fig.

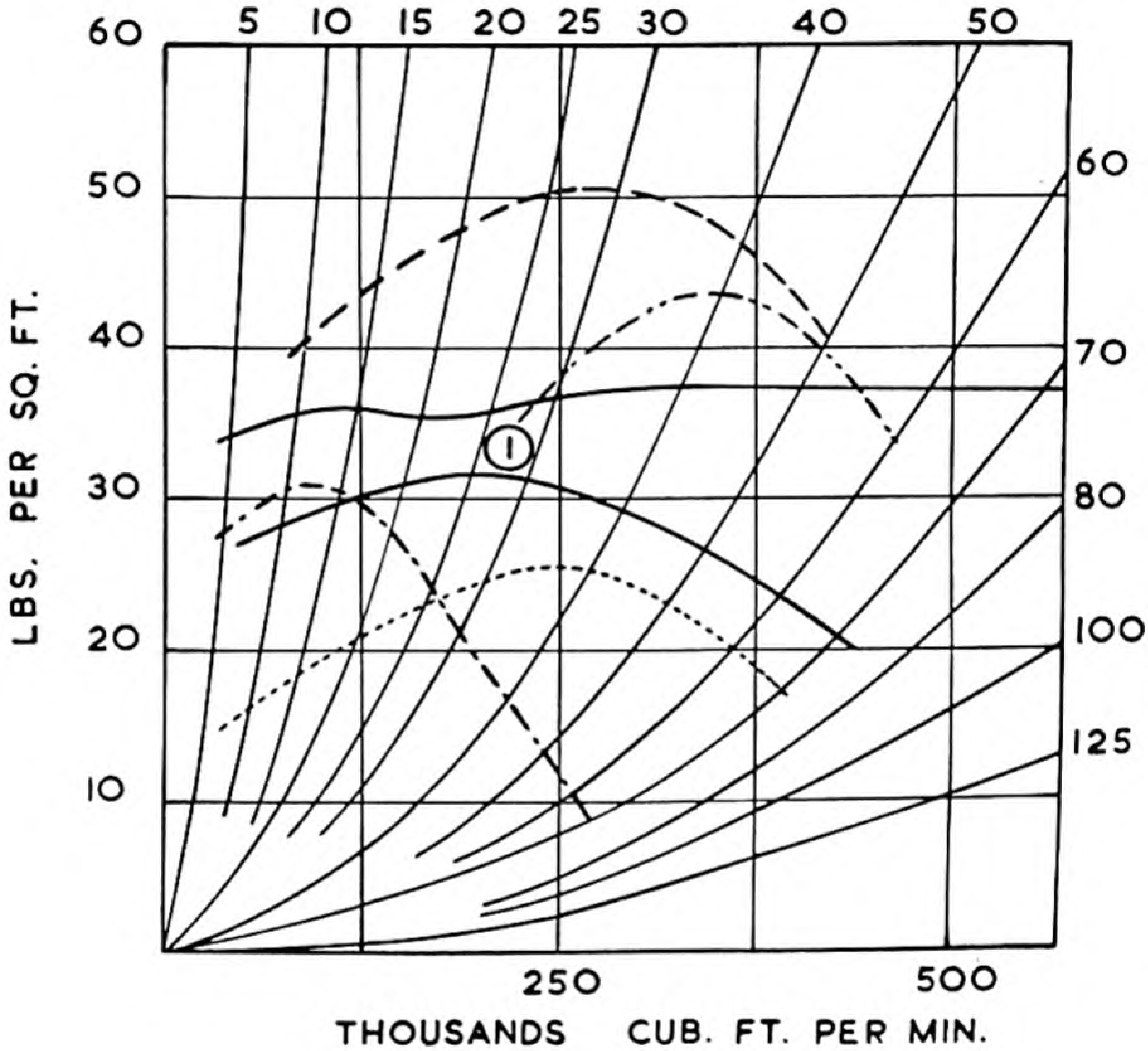


FIG. 383

383, rising to a peak and falling again with increasing quantity. Thus, two quantities may be obtained for each pressure difference, giving a large and a low value. The lower quantity is related to the smaller equivalent orifice and vice versa. Only a few fans are so designed to provide a flat characteristic, in which case the curve may be a straight line. In such cases the fan will produce different quantities with approximately the same pressure difference. The characteristic of another fan type falls steadily with increasing quantity and equivalent orifice.

An important factor is the point on the family curve at which the

fan will run at a definite speed, i.e. to which pressure and quantity the fan will adjust. The fan speed will depend upon the design of the fan itself, expressed in relation to its family curve and the family curve of the mine or the equivalent orifice. The actual point of the intersection of these two curves determines the operation point of the fan. A fan with a family curve 1 in Fig. 383 operating against an equivalent orifice of 30 square feet will pass a quantity of 230,000 cubic feet per minute against a pressure of 31 lb. per square foot. If the equivalent orifice is increased to 50 square feet, the pressure difference decreases to 27 lb. per square foot and the quantity increases to 340,000 cubic feet per minute. It is extremely important to ensure that the fan is operating within the most favourable efficiency range. Since the efficiency depends upon the equivalent orifice, it is obvious that each point on the family curve has a definite fan efficiency. In the case of the normal centrifugal fan, the best efficiency is generally just below the peak for the fan. Should this point coincide with or be close to the operating point for the fan, the ventilation and fan efficiencies will be high. If the operating-point is outside this range, the fan efficiency will be moderate or poor. If the reduced pressure difference resulting from a large equivalent orifice is considered in relation to the low fan efficiency, the saving in power may compensate for the loss in efficiency. This position does not frequently arise, and it is considered that a fan operating at a low efficiency has been badly chosen in the first instance. Temporary deviations from the operating-point may occur due to the continual change in the equivalent orifice as a mine develops. In this way it is reasonable to assume that during the life of a mine the efficiency will gradually deteriorate if the same fan is continuously employed.

In the P–Q diagram in Fig. 384, the family curves have been drawn for a screw fan running at four different speeds. The equivalent orifice family curves and the fan efficiency curves have also been added.

The highest fan efficiency of 75 per cent. is obtained with an equivalent orifice of 27 square feet. The high efficiency range is very wide, and it does not drop below 70 per cent. even with an orifice range of from 6 to 15 square feet. The figure clearly shows that the efficiency tends to fall and that the high efficiency range narrows with a decrease in the fan speed.

Another form of representation of fan characteristics is shown in Fig. 385(a). The quantity, corresponding pressures and fan efficiency, as well as the motor b.h.p. and a.h.p. (air horse-power), have been plotted. The figure shows that the best efficiency has been reached with an equivalent orifice of 20·5 square feet. The fan operates with an efficiency of only 46 per cent., however, when the equivalent orifice is 37·7 square feet instead of the maximum of 54 per cent. In view of these characteristics, each subsequent increase in the equivalent

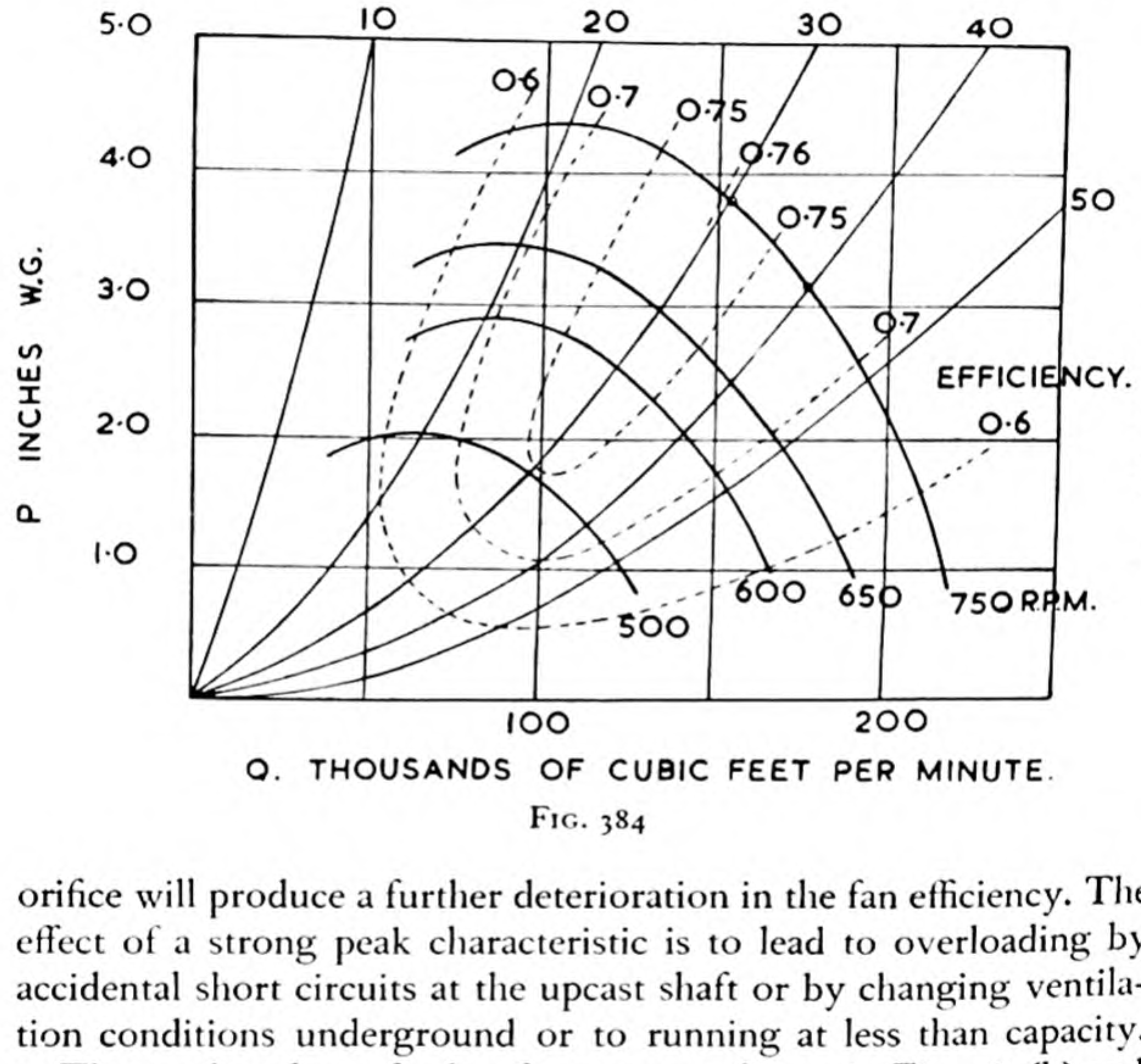


Fig. 384

orifice will produce a further deterioration in the fan efficiency. The effect of a strong peak characteristic is to lead to overloading by accidental short circuits at the upcast shaft or by changing ventilation conditions underground or to running at less than capacity.

The results of two further fan tests are shown in Fig. 385(b) and Fig. 385(c). The former relates to the main fan installation and the latter to the reserve or stand-by fan. It can be seen that the rating or duty of the main fan installation is wrong. The fan is working with an efficiency of only 38 per cent. at an orifice of 31 square feet, since it has been designed to operate on a much larger orifice. The older reserve fan, however, has been designed to operate on an orifice of 28 square feet and would operate with an efficiency of about 61 per

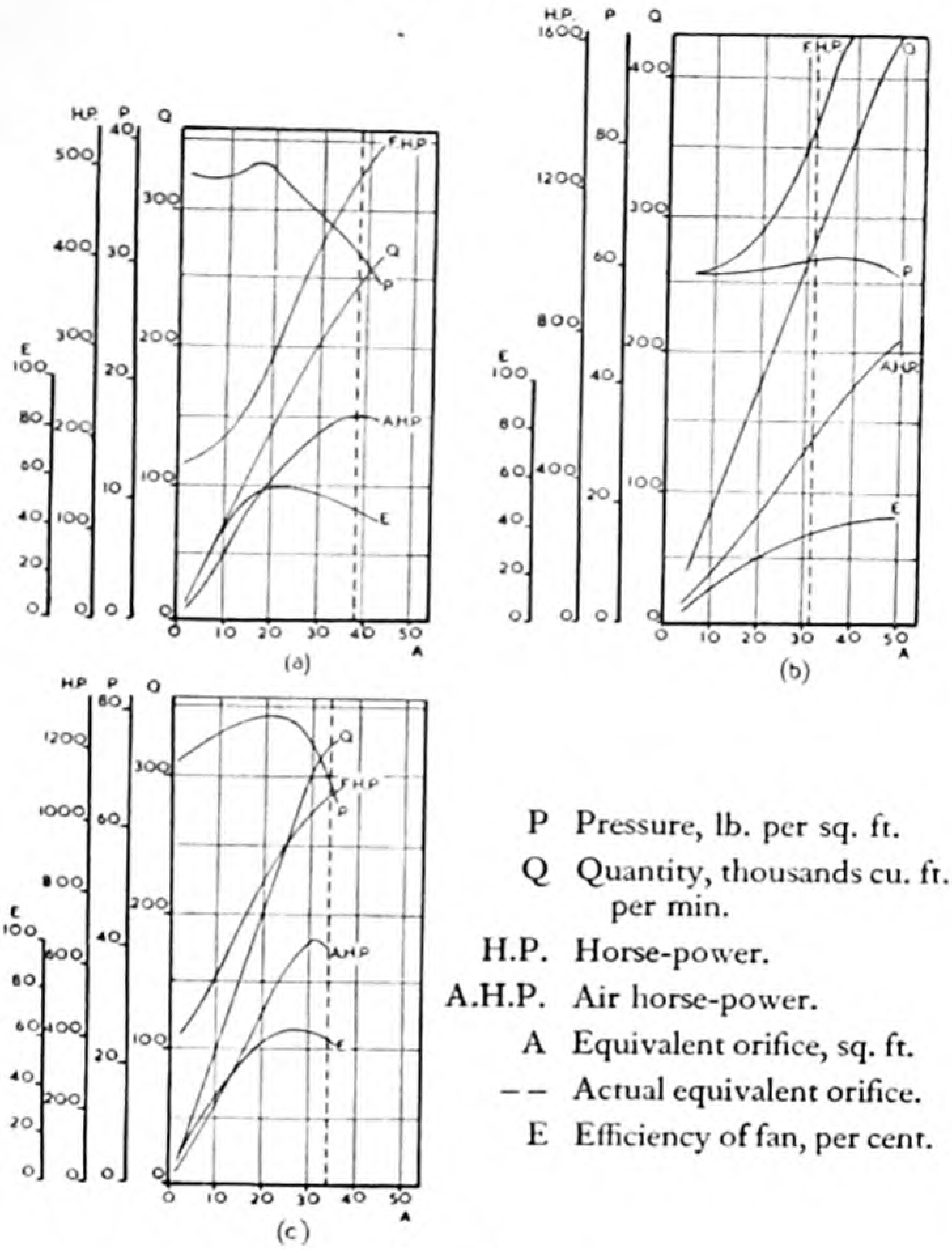


FIG. 385

cent. at an equivalent orifice of about 34 square feet. This fan requires 200 kW. less than the main fan, and at a power cost of 1d. per kWh. its operation would mean a saving of about £7,300 per annum. It is necessary carefully to distinguish between the scope of the best fan-efficiency and the much more extended working range of the fan. The latter is determined by a lower and upper value of the equivalent orifice and the maximum possible speed of the fan. The lower limit is given by the pumping limit of the fan, which is shown in P–Q diagrams, where the family curve for the fan runs parallel to the equivalent orifice curve, e.g. in Fig. 384 at an area of 17 square feet. The upper limit depends upon the orifice of passage of the fan. The maximum fan speed is limited by the design of the fan and drive.

PART V

THE EFFICIENCY OF THE VENTILATION SYSTEM

A distinction must be made between the fan efficiency and the efficiency of the underground ventilation system. The latter may be represented by the ratio of the capacity utilised in the mine to the capacity of the fan, i.e. $E_m = \frac{Q_m}{Q_f}$. It has already been noted in a previous section that only part of the downcast air is utilised at the coal face, the remainder being used in other circuits or dissipated through leakage or restrictions in the airways. The proportion of the fan output which is usefully applied at the coal face is usually between 20 and 70 per cent., the average being 40 per cent. The air horse-power absorbed by the actual ventilation splits is less than that used to provide for leakage and other useless circuits. After deducting these losses, the real air horse-power demand can be calculated for the actual working areas, while the pressure required can be computed by referring to the roadway resistance values.

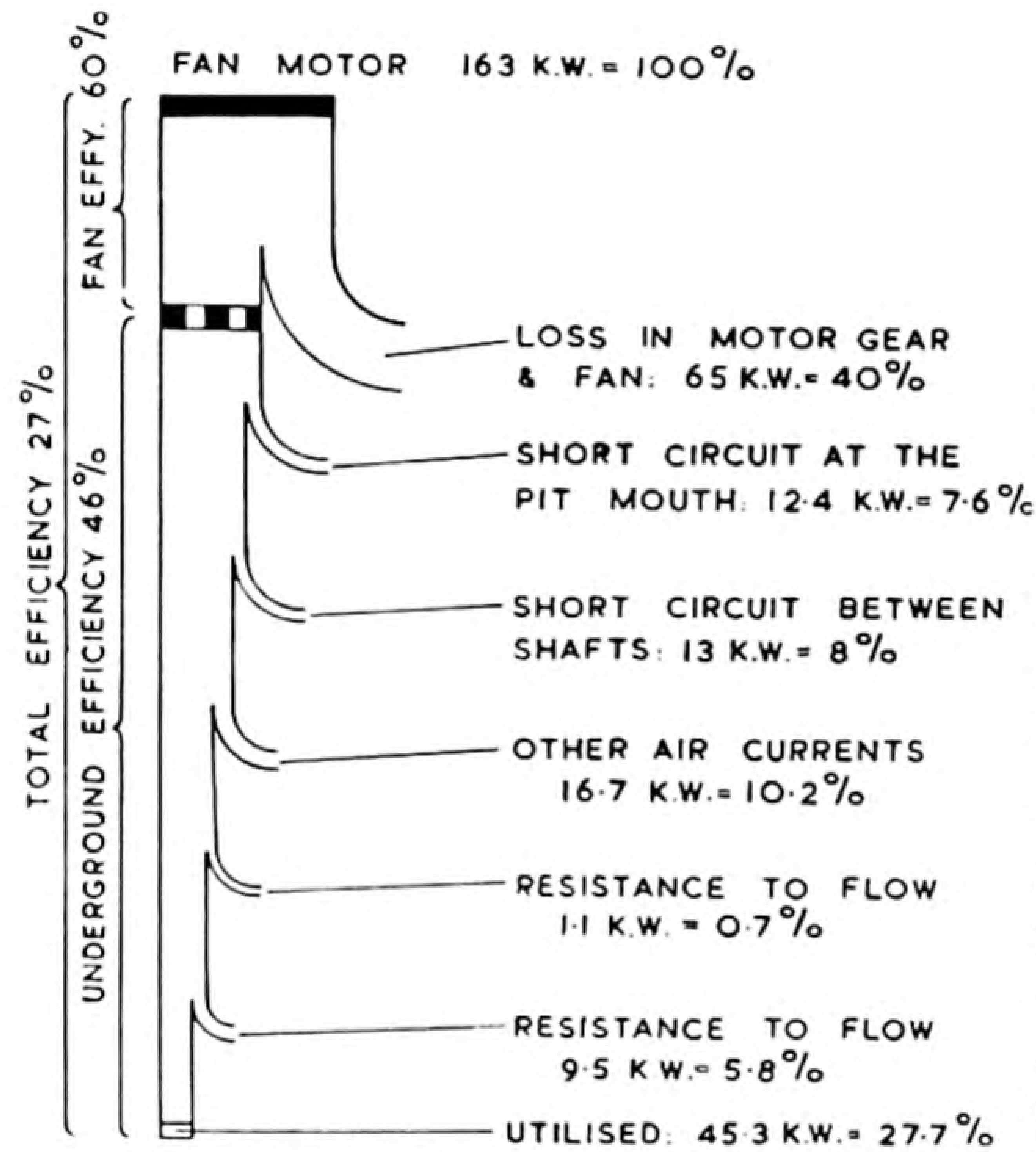


FIG. 386

The overall efficiency of the mine ventilation system can be determined if this calculation includes the fan efficiency and the efficiency of the underground ventilation system. Since the fan efficiency averages about 60 per cent. and the underground efficiency about 40 per cent., the total, or overall, efficiency is in the region of 25

per cent. In certain favourable cases it may rise to 50 per cent., and in other poor systems it may fall as low as 5 per cent.

These values will only indicate where the major fall in efficiency occurs. The reasons for low efficiences can be explained only by detailed analysis. One method is shown in Figs. 386 and 387, in which Sankey diagrams are used to demonstrate the source of the

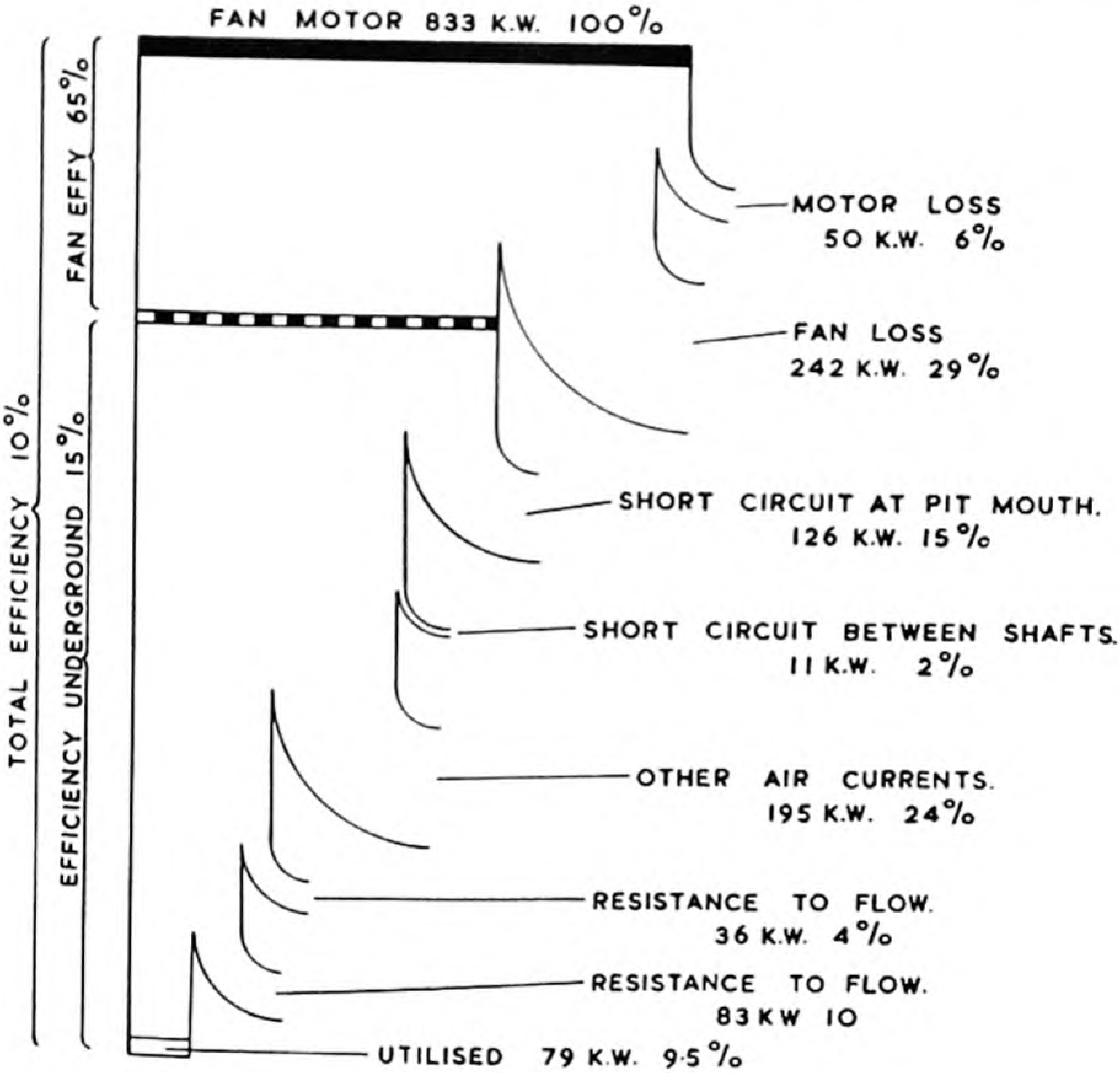


FIG. 387

losses. In the case shown in Fig. 386, the fan efficiency is good and the underground and overall efficiencies are above the average. In the case shown in Fig. 387, the fan efficiency is very good, the underground ventilation system, however, is poor, so that the overall efficiency is low. The high leakage value at the upcast shaft top exceeds all other leakage losses.

PART VI
AUXILIARY VENTILATION

Section 1. General Introduction

Auxiliary ventilation by underground fans is indispensable for development headings either in coal or in rock where the headings cannot be connected directly to the main ventilation system. The development work may include cross-cuts, conveyor gates, rises and staple shafts, which may be driven upwards or downwards. The development of stone drifts in advance of coal development will generally require an auxiliary ventilation installation adequate to ventilate the heading(s) well in advance of the main ventilation circuit. The installation of auxiliary fans must be carried out in accordance with regulation No. 14 of the Coal Mines (Ventilation) Gen. Regs. No. 974 (1947). Proper installation and maintenance of such fans and the fan ducting is imperative in order that the quantity required can be constantly delivered at the face. Sheet-metal ducts are used extensively for more permanent installations; canvas ducting is, notwithstanding its higher resistance, also employed since it can easily be made airtight, is resistant to dampness and is easy to handle when installing and during transportation.

Section 2. The Air Tubes or Ducting

The circulation of fresh air to the face of an advancing heading, rise or staple shaft requires the provision of separate intake and return circuits. In the case of an auxiliary fan installation, the ducting and the roadway or rise will fulfil one or other of these duties. In the case of a forcing fan installation, the intake air will travel via the ducting to the face while the roadway serves as the return, whereas in an exhaust fan installation, the ducting will serve as the return.

The ducting may consist of welded steel tubes with a wall thickness of from $\frac{3}{64}$ to $\frac{5}{64}$ inch. The tubes are usually galvanised as a protection against rust or corrosion. Asphalt-covered or black lacquered tubes have also been used. The tube diameter varies from 12 to 36 inches. In rises and staple shafts the tube diameter is usually from 12 to 16 inches, while in stone drifts ducting up to 24 inches diameter is used. The larger-diameter tubes are used only in exceptional circumstances where the methane emission is high and large

quantities of air must be passed. Each tube section should be as long as possible so as to eliminate leakage losses at the connections. Lengths of 10 feet are usual, any greater length being difficult to transport.

It is important to reduce the pressure and quantity losses in the duct line to a minimum. The efficiency of the ducting can be defined as the relationship between the product of the required quantity at the inbye end of the duct and the pressure needed to transport this quantity in an airtight line, and the product of the actual quantity moved by the fan and the actual pressure absorbed. The pressure loss in the line will depend upon the frictional resistance of the

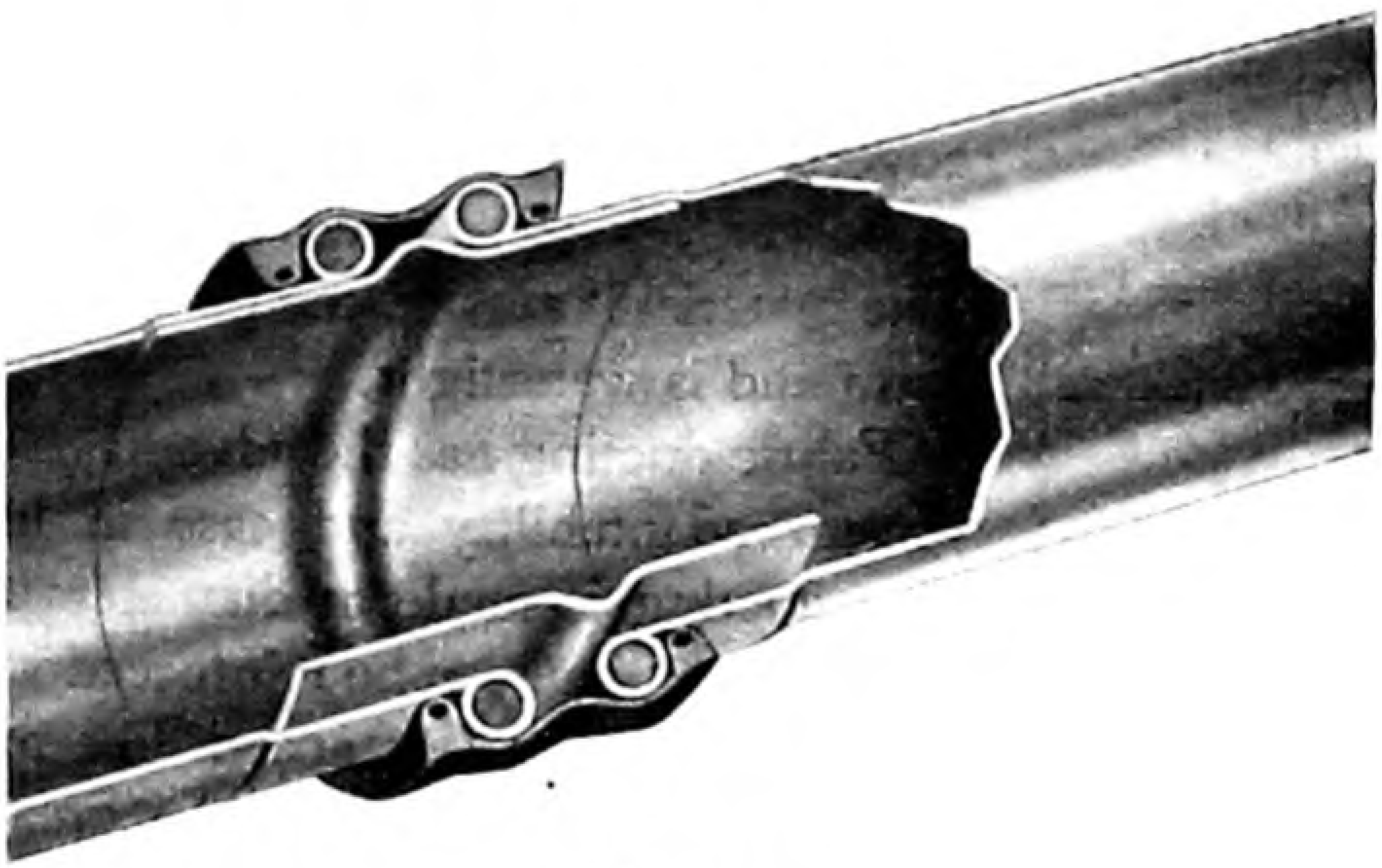


FIG. 388

tubes. The roughness of the tubes will play only a small part, the greater losses occurring at the tube connections. The efficiency of the tube connections is also a decisive factor in reducing leakage losses. The joint should not reduce the duct sectional area, and invariably special connections are used. The higher cost of these special joints is more than compensated by the saving in power at the fan. A German type, made by Brandt (Duisburg), is illustrated in Fig. 388. This joint consists of an annular intermediate galvanised steel ring, over which the adjoining tubes slide. One of the tubes is provided with a rubber sleeve fitting over the circular reinforced collar at the end of the tube. The joining tube is inserted into the rubber sleeve over the intermediate steel ring so that its reinforced collar is retained in the rubber sleeve. This joint is very flexible, and

curves can be laid up to 8 degrees deflection. A type of tubing for vertical suspension in shafts also made by Brandt is shown in Fig. 389. The connection is in the form of a bayonet joint and, like the previous type, has an intermediate section and rubber rings. The joint is said to take a load of 7,000 lb. With the normal type of flanged or spigoted joint, the efficiency decreases considerably with extended use, due to damage, caused by the continual repetition of fitting and removal by transport and shot firing. One method of improving these joints is the use of the Denso Bandage manufactured by the Chemieprodukt G.m.b.H. Bergneustadt. This bandage, which has been used mainly as a protective covering for cables and pipes, has been found to be very effective in covering poor joints. The material is plastic, can be adapted to any form and is watertight.

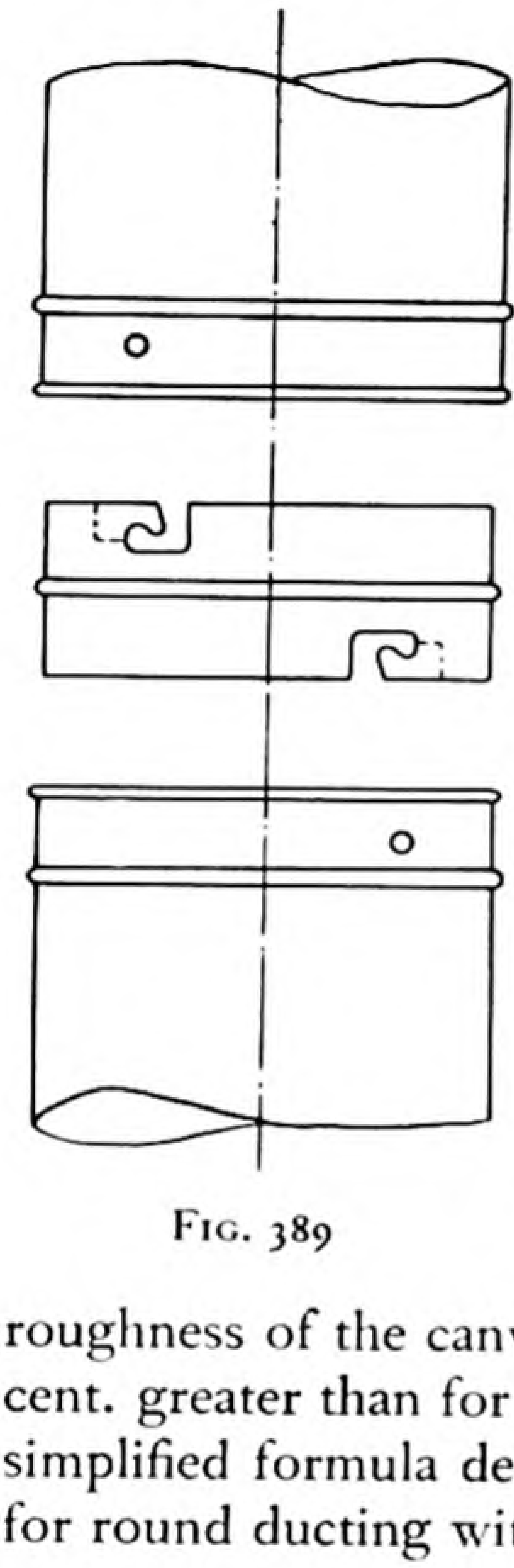


FIG. 389

Fabric ducting has the advantage of much greater adaptability and ease in handling. The ducting is made in diameters from 10 to 30 inches, and can be reinforced with steel rings set about 2 feet apart. The resistance of canvas ducting due to the greater wall roughness of the canvas compared with steel sheet is about 40 per cent. greater than for steel tubes. MacFarlane* gives the following simplified formula derived from the Fanning friction-loss formula for round ducting with no joints:

$$H_f = C\frac{L}{D}h_v,$$

where H_f is the friction loss in inches water gauge, L the length of the duct in feet, D is the diameter of duct in feet and h_v the velocity head in inches water gauge. The velocity head can be obtained from the relationship—

$$h_v = \left(\frac{\text{vel. in duct ft./min.}}{3{,}970}\right);$$

* *Ventilation Engineering*, D. MacFarlane. Davidson & Co. Ltd., Sirocco Works, Belfast.

while C is a constant which is 0·02 for circular steel ducts and 0·028 for canvas ducting. The nomogram by MacFarlane in Fig. 390 shows the resistance values for steel and canvas ducting based upon a difference of 40 per cent.

German experience is that the difference between steel ducting

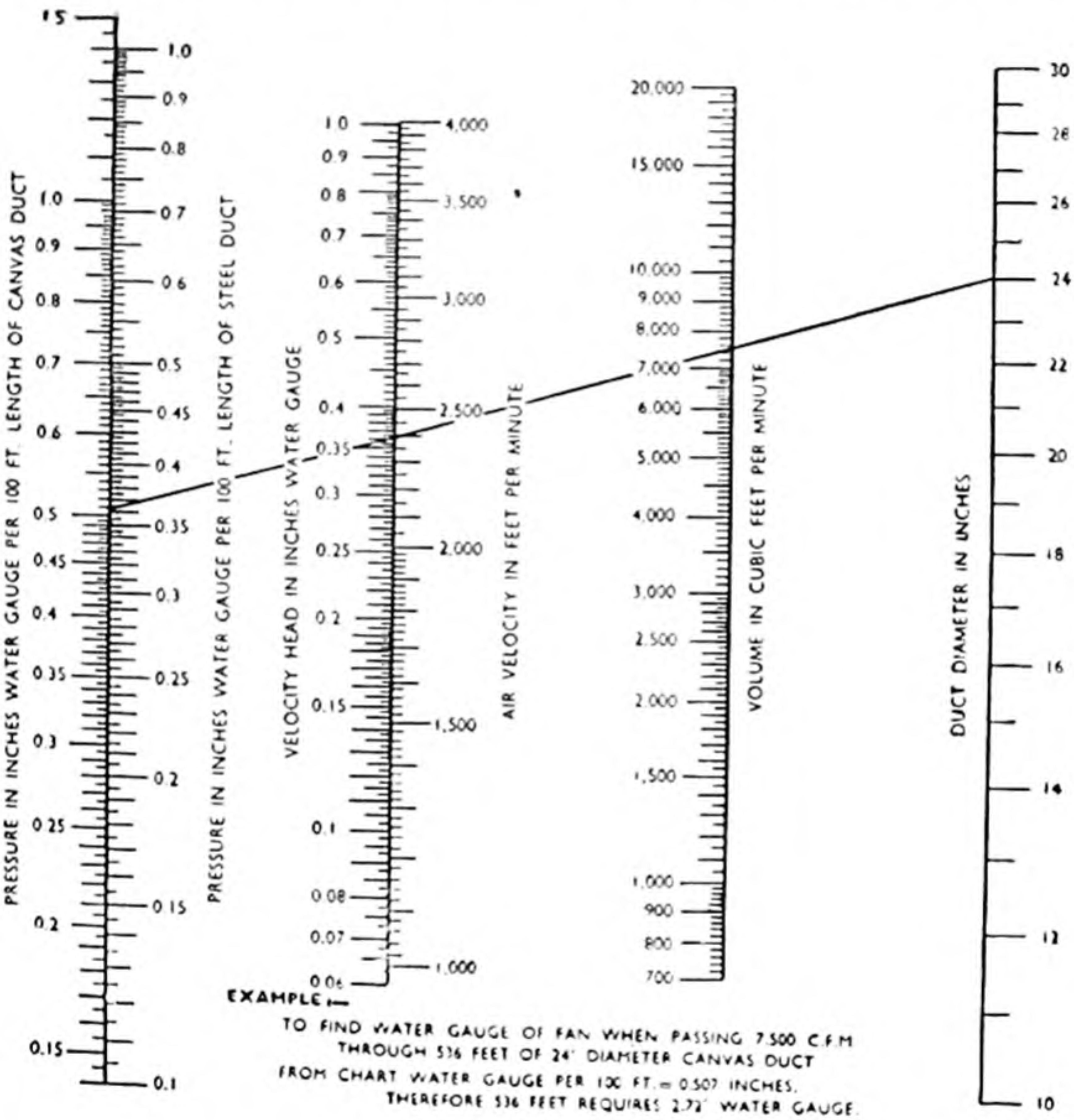


FIG. 390

and canvas ducting is about 150 per cent. instead of 40 per cent., and in Germany canvas ducting is used only for distances of up to 60 yards. It is especially suitable for the end of a duct of steel tubes, because it can easily be extended to the face when men are at work and be pushed back during shot firing. The end can be fitted with a pulley-sheave, which facilitates advance after shot firing. With the canvas duct, only forcing ventilation can be applied.

The canvas ducting can be suspended on a wire slung from the

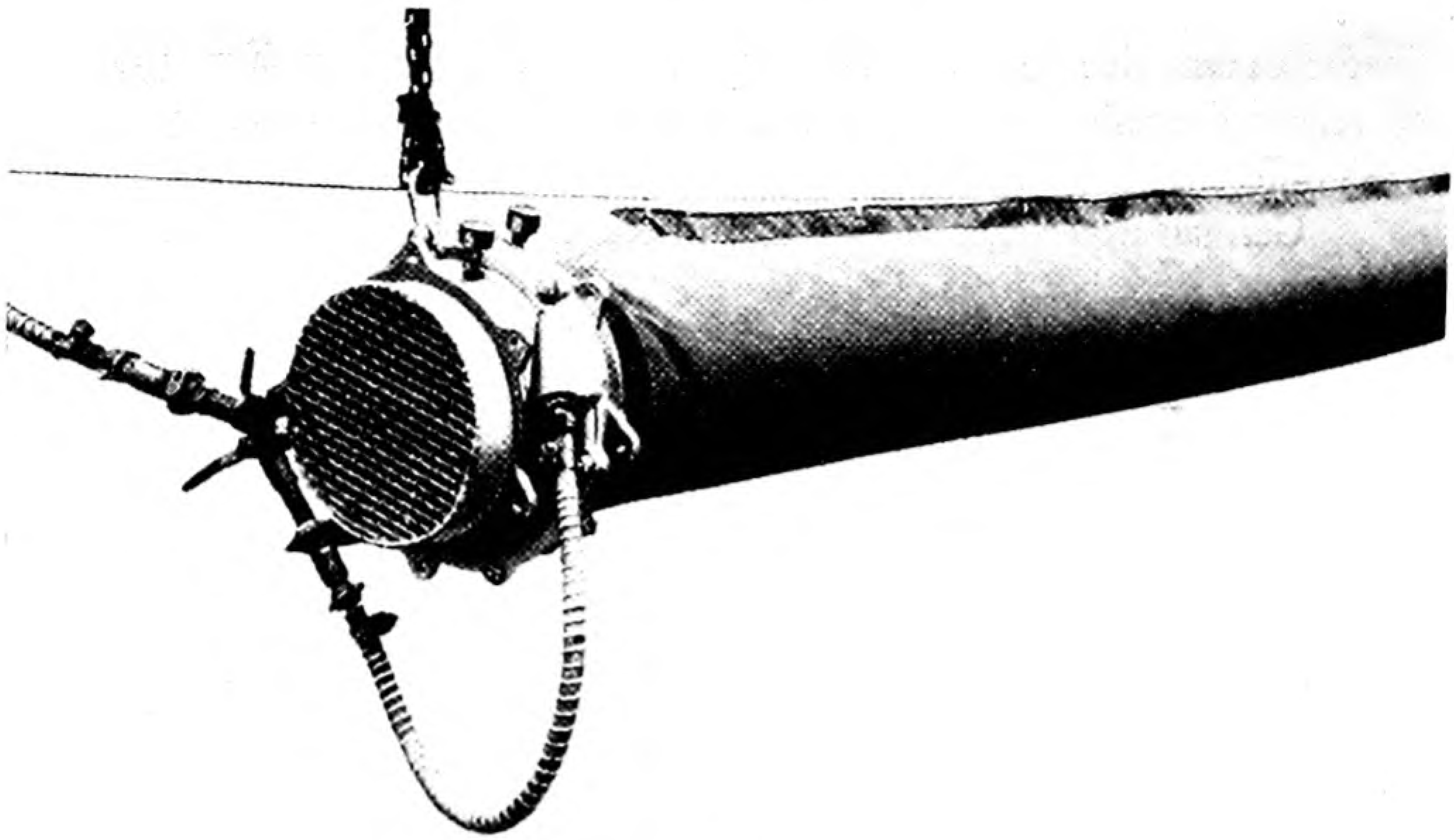


Fig. 391

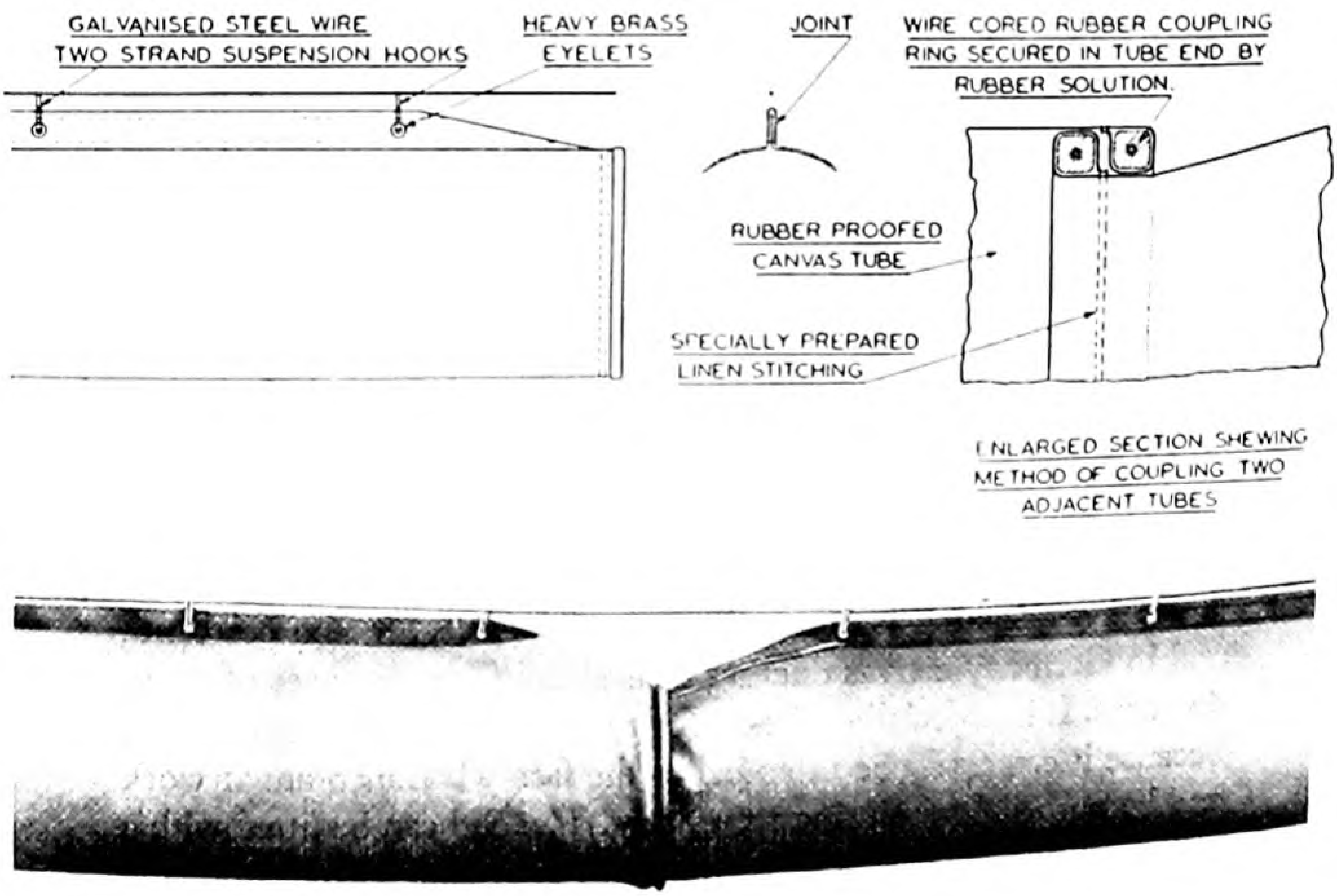


Fig. 392

roof as shown in Fig. 391, the tubing being provided with small suspension hooks on to the wire. Flexible fittings can be provided to cover any particular installation, including reducers to accommodate smaller-diameter tubing from a larger-diameter fan. The method of coupling one type of canvas ducting is shown in Fig. 392. Where the tubing is used for shaft ventilation, special suspension collars are provided every 100 feet to support the tube from the shaft cribs.

Section 3. Auxiliary Fans

The propeller or screw fan is more widely adopted than the centrifugal for duct ventilating systems, due to its flat or non-over-

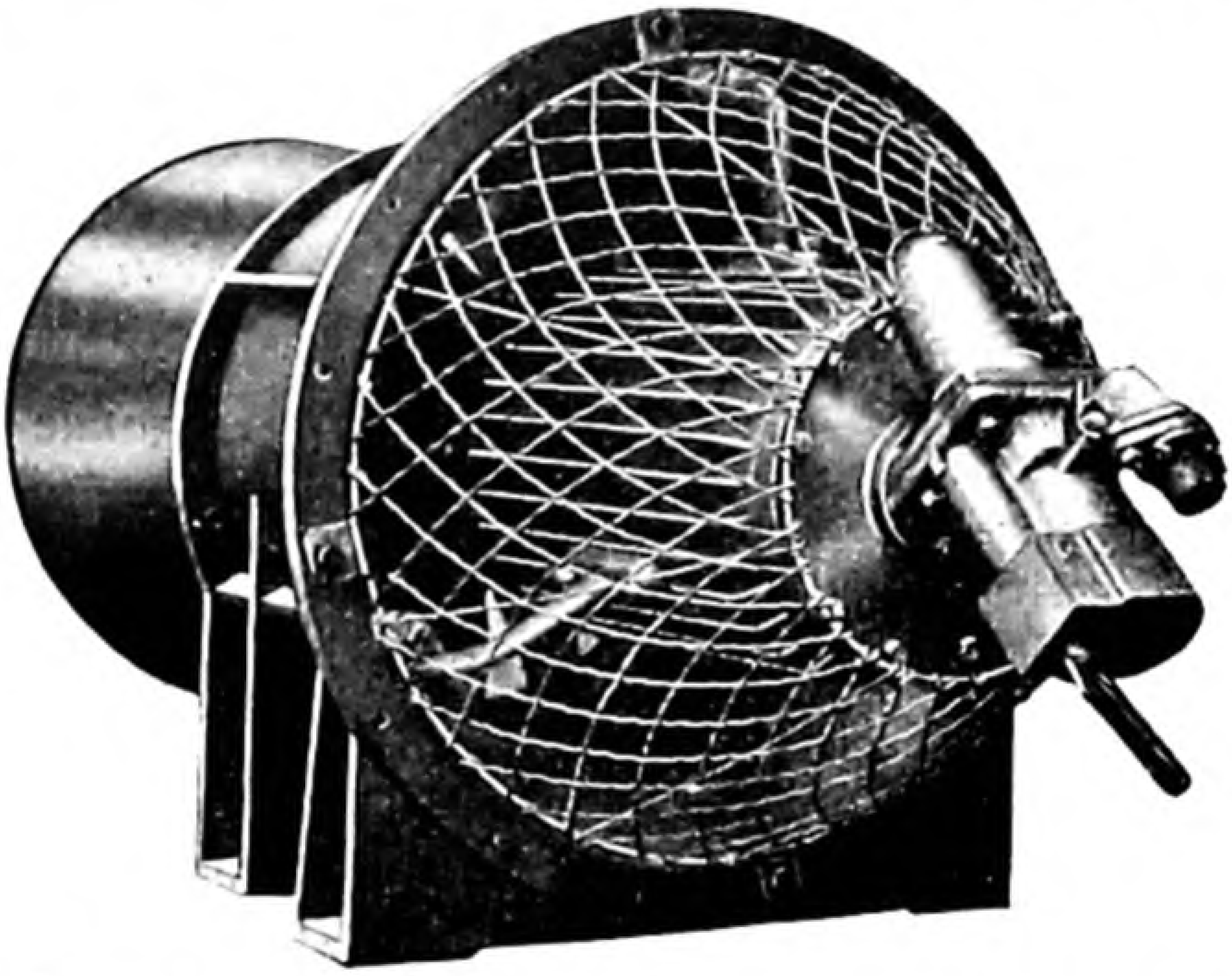


Fig. 393.—20 in. diameter 'Aeroto' auxiliary fan, showing flame-proof motor and terminal plug.

loading power characteristic. The axial flow fan is smaller in size than the centrifugal fan and can be more easily accommodated in the roadway. The fan can be coupled directly in line with the ducting, and in some types the drive is direct and integral with the fan casing. The fans may be driven by either electricity or compressed air and can be coupled in series in a long duct line. The auxiliary fan has to be capable of providing a high overall efficiency and operating over a wide range of duties, since it is delivering air against the varying resistance of an increasing length of ducting. The axial-flow fan has not the steeply rising power characteristic of the centrifugal fan, which, if operated on a large equivalent orifice, such as would be

provided by a short length of ducting on the exhaust side, would seriously overload the motor drive.

A 20-inch diameter 'Aeroto' auxiliary fan is shown in Fig. 393. This fan is direct-driven by a 5-h.p. A.C. flame-proof motor and is capable of supplying from 5,000 to 10,000 cubic feet per minute on a varying length of canvas duct up to 1,400 feet. The performance curves for this fan are shown in Fig. 394.

An interesting fan of German design, manufactured by Nüsse and Gräfer, is shown in Fig. 395. This fan is provided with a handwheel in the centre of the rotor boss, by which the pitch of the rotor blades may be altered to regulate the fan capacity. The fan shown in Fig. 396, with an external drive, is provided with two stages, and can operate up to 20-inch W.G. One stage can be omitted when the fan is operating on a shorter duct line, and double-stage operation introduced when the length of the ducting has been considerably increased. Since the fan motor is external, the fan can be more readily used for exhaust ventilation in gassy mines than is the case with direct-driven fans.

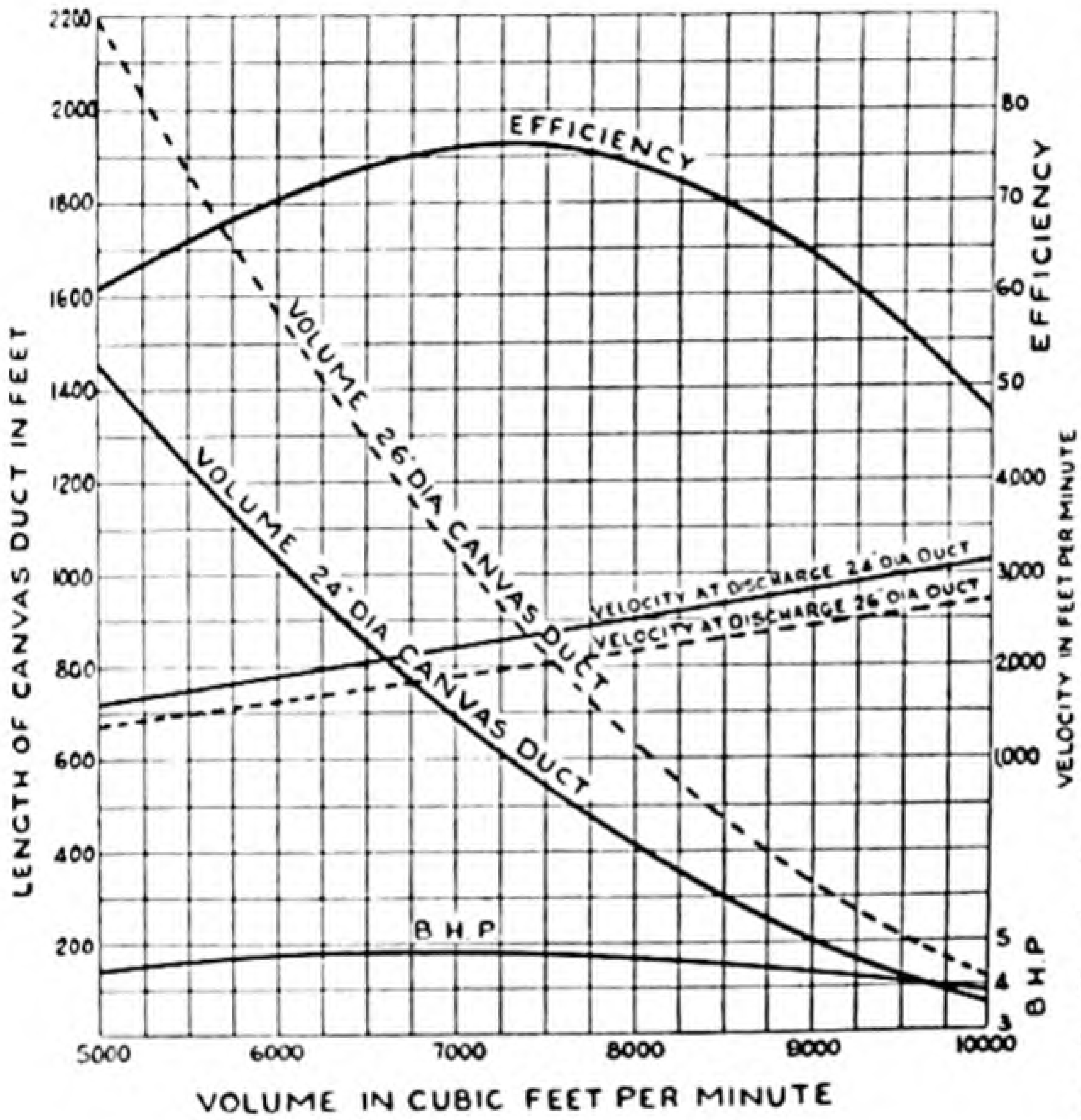


FIG. 394

Another development to give a measure of volume control for face requirements is the Meco E.F.5. direct-driven auxiliary fan shown in Fig. 397. This 20-inch-diameter fan is driven by a 3,000 r.p.m., synchronous, squirrel-cage, flame-proof motor, and is provided with a variable volume control device in the form of steel butterfly wings mounted on a central spindle and externally operated by a rust-proofed screw and nut. The device can be

operated without stopping the fan, and is useful in regulating the delivery from fans in series so that tube collapse is obviated. The performance curves for this fan are shown in Fig. 398.

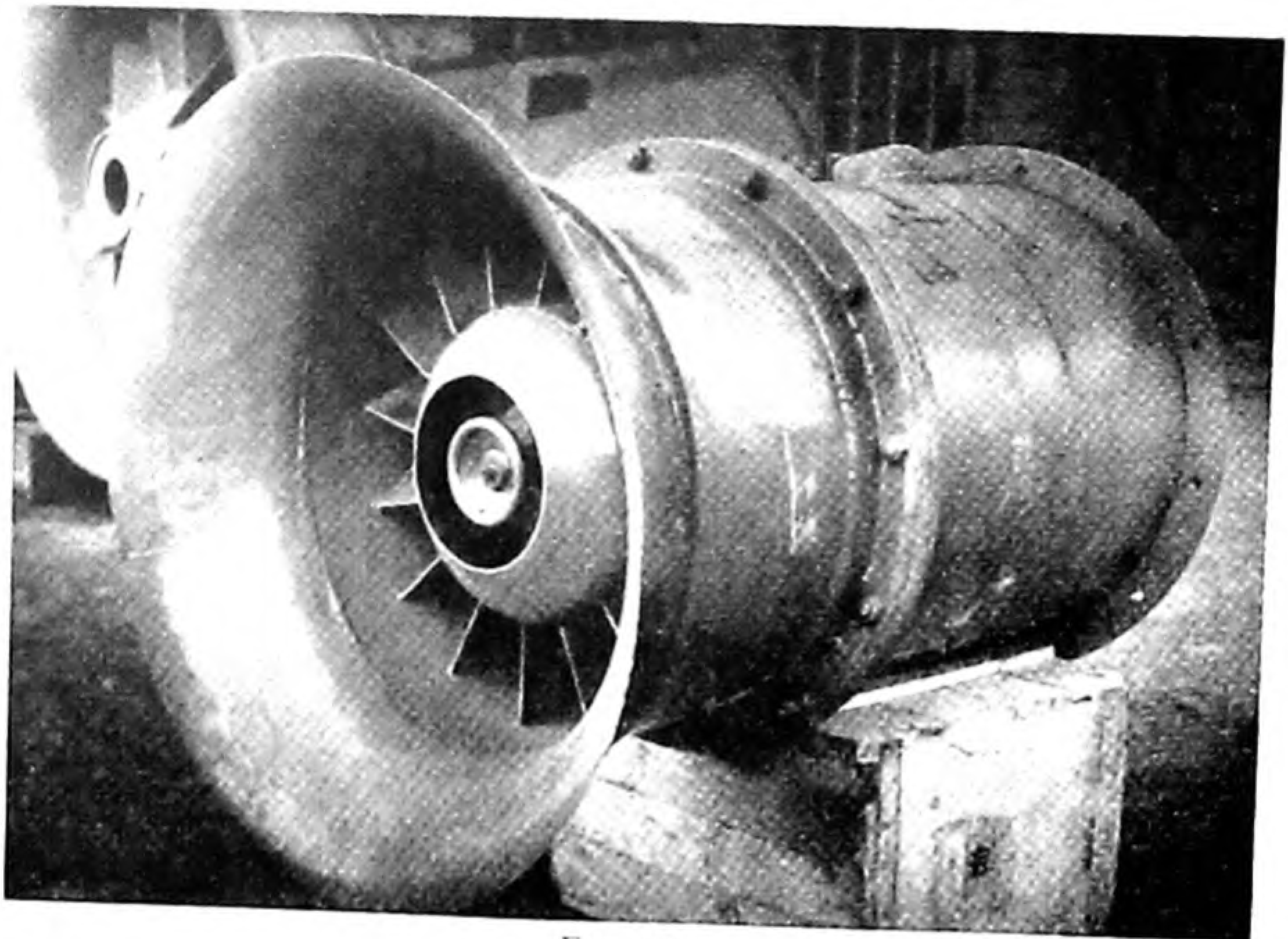


FIG. 395

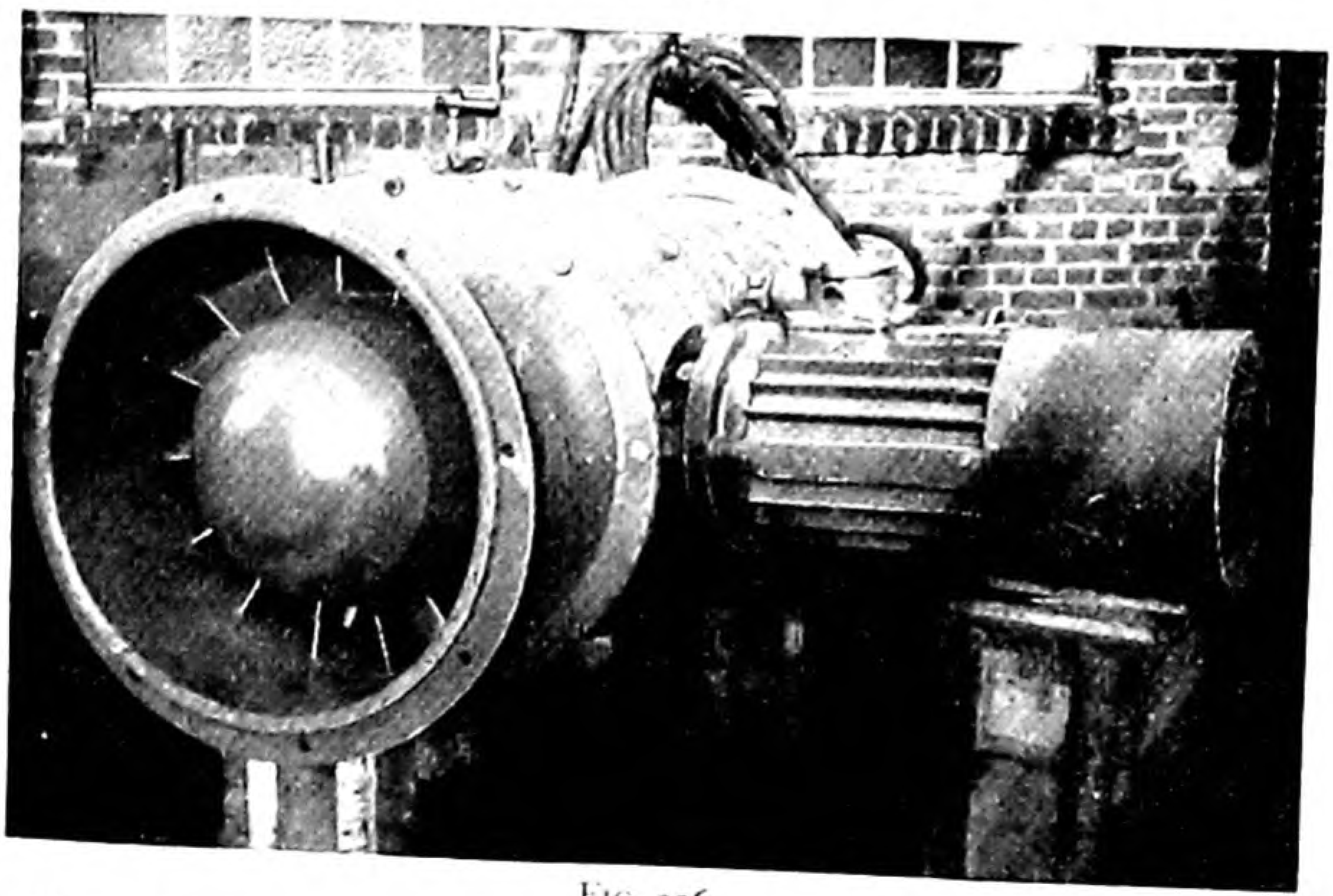
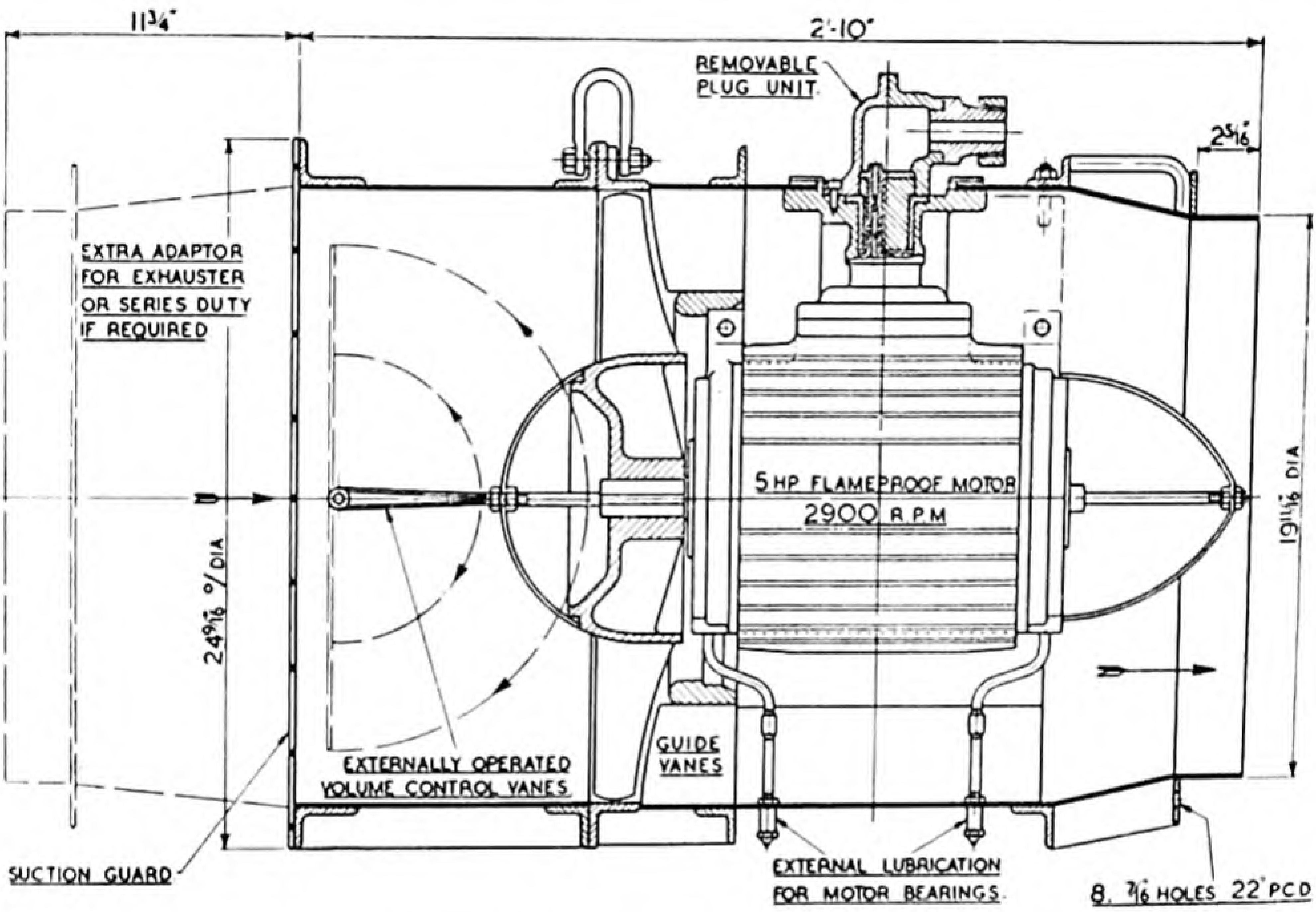


FIG. 396

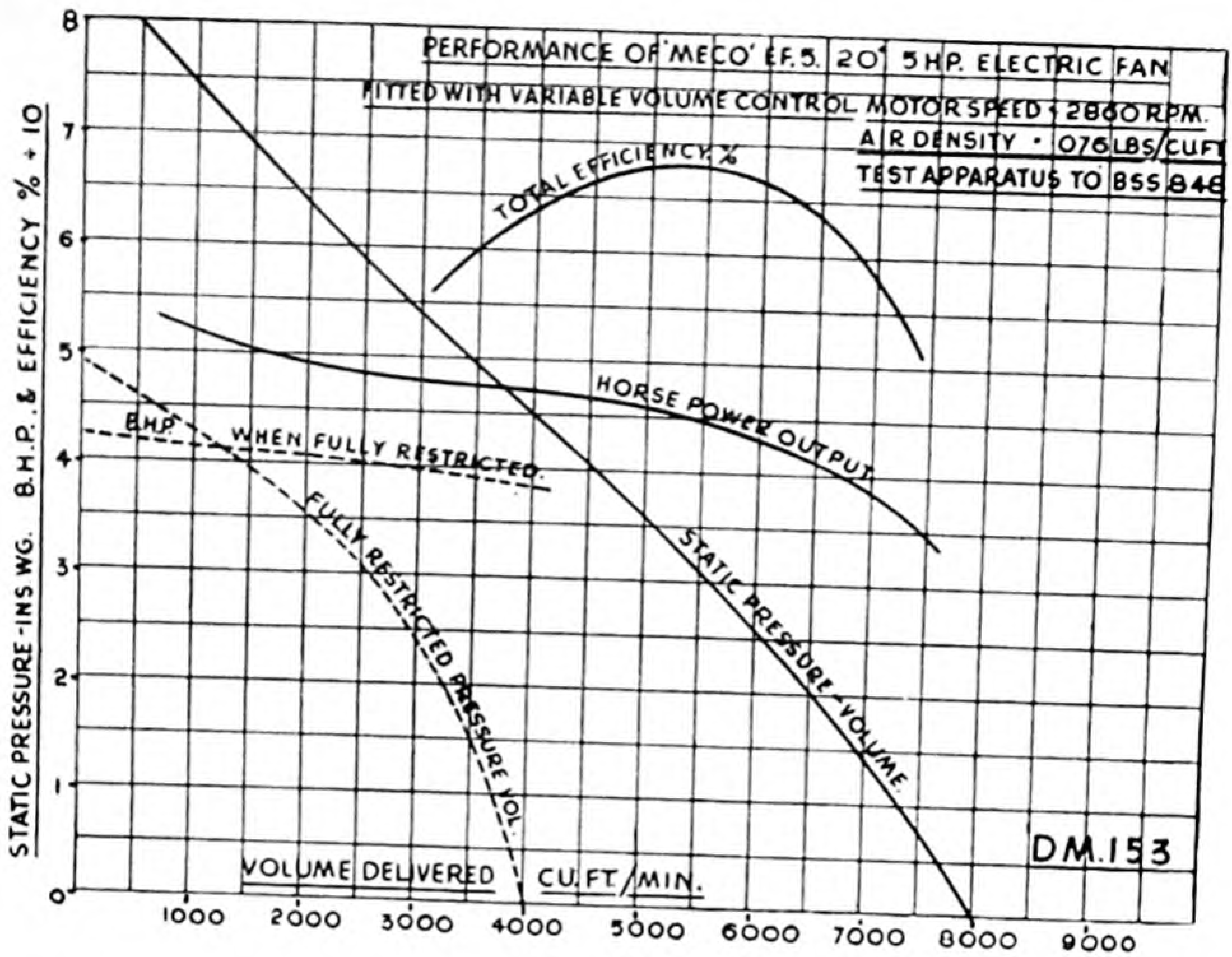
An interesting fan design has been developed by Woods of Colchester, in which the contra-flow principle is utilised. The aerofoil auxiliary fan is made in varying sizes, but the fan designed for underground work is 15-inches diameter, as shown in Fig. 399. The first and second stage impellers are of the adjustable-pitch pattern, each stage being independently driven by a direct motor drive. The impellers are gravity die-cast from silicon aluminium, which makes



SECTION OF E.F.5. ELECTRIC FAN.

FIG. 397

for lightness and high resistance to corrosion. A range of static pressure over 5 inches water gauge is possible with this fan. The rotor speed is 2,800 r.p.m., the motor developing a maximum b.h.p. of 3·9. The fan is capable of passing 3,000 cubic feet per minute through 16-inch ducting at 1,000 feet from the fan, and giving a peak efficiency of over 70 per cent. on total water gauge and 63 per cent. on static gauge. The noise level with this type of fan is much less than that produced by a single-stage fan producing an equivalent volume, since the tip speed is 60 per cent. less and noise is proportional to the impeller tip speed.


PERFORMANCE OF 'MECO' E.F.5. 20" 5 H.P. ELECTRIC FAN
FITTED WITH VARIABLE VOLUME CONTROL MOTOR SPEED 2860 R.P.M.
AIR DENSITY · 076 LBS/CU.FT.
TEST APPARATUS TO BSS 848
TOTAL EFFICIENCY %
HORSE POWER OUTPUT
B.H.P. WHEN FULLY RESTRICTED
FULLY RESTRICTED PRESSURE VOL.
STATIC PRESSURE-VOLUME
STATIC PRESSURE - INS WG. B.H.P. & EFFICIENCY % ÷ 10
VOLUME DELIVERED CU. FT./MIN.
DM.153
0
1
2
3
4
5
6
7
8
1000
2000
3000
4000
5000
6000
7000
8000
9000

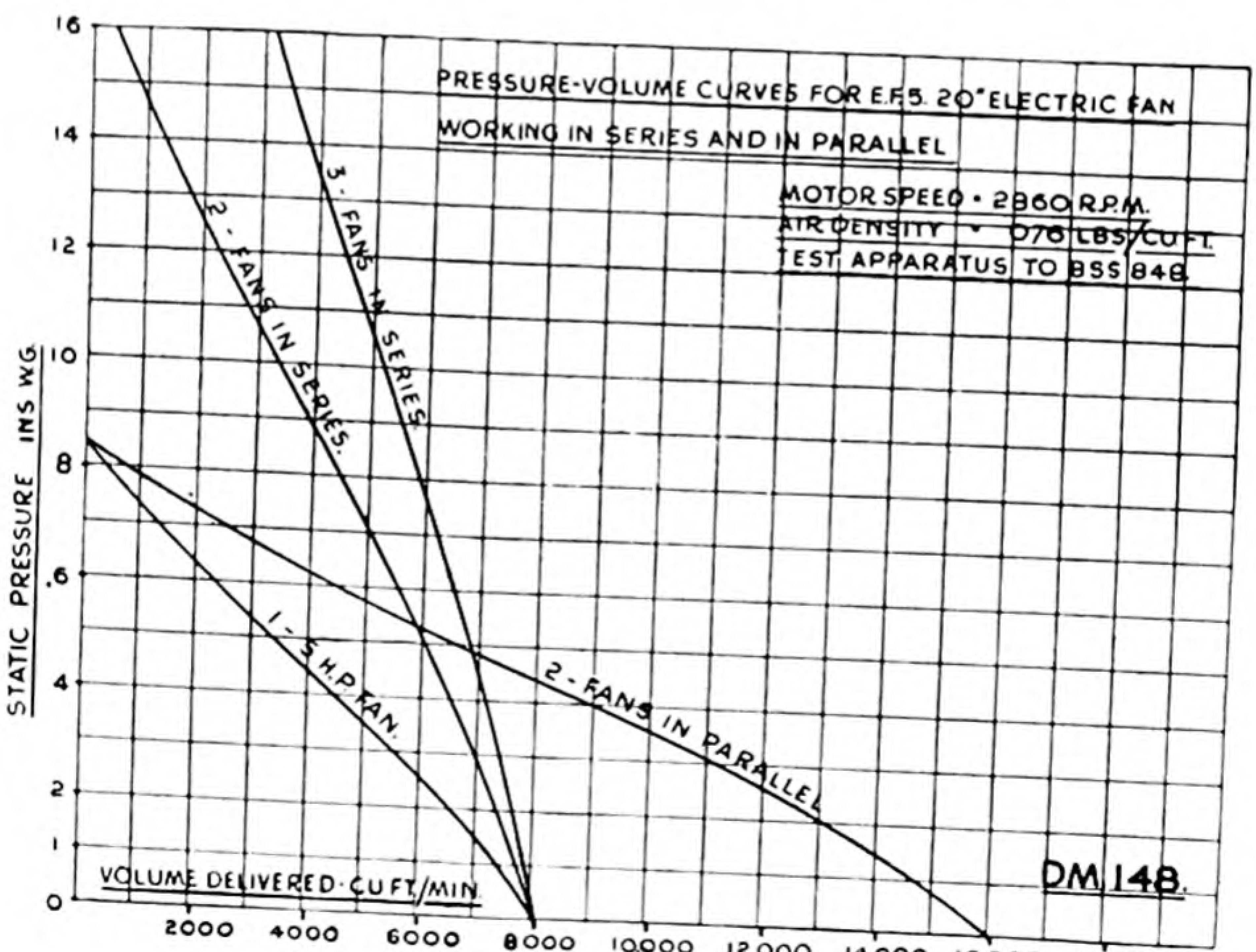

PRESSURE-VOLUME CURVES FOR E.F.5. 20" ELECTRIC FAN
WORKING IN SERIES AND IN PARALLEL
MOTOR SPEED · 2860 R.P.M.
AIR DENSITY · 076 LBS/CU.FT.
TEST APPARATUS TO BSS 848
3 - FANS IN SERIES
2 - FANS IN SERIES
1 - 5 H.P. FAN
2 - FANS IN PARALLEL
STATIC PRESSURE INS WG.
VOLUME DELIVERED · CU FT/MIN.
DM.148.
0
1
2
4
6
8
10
12
14
16
2000
4000
6000
8000
10,000
12 000
14.000
16.000
18.000

FIG. 398

Compressed-air-operated auxiliary fans are generally turbine driven, the fan rotor and turbine being combined. The sectional drawing in Fig. 400 shows the constructional details of a turbine fan and illustrates the nozzle and turbine blade design. This unit is driven by two air jets impinging on the turbine blading. The blades are dovetailed into the run of the rotor and locked in place

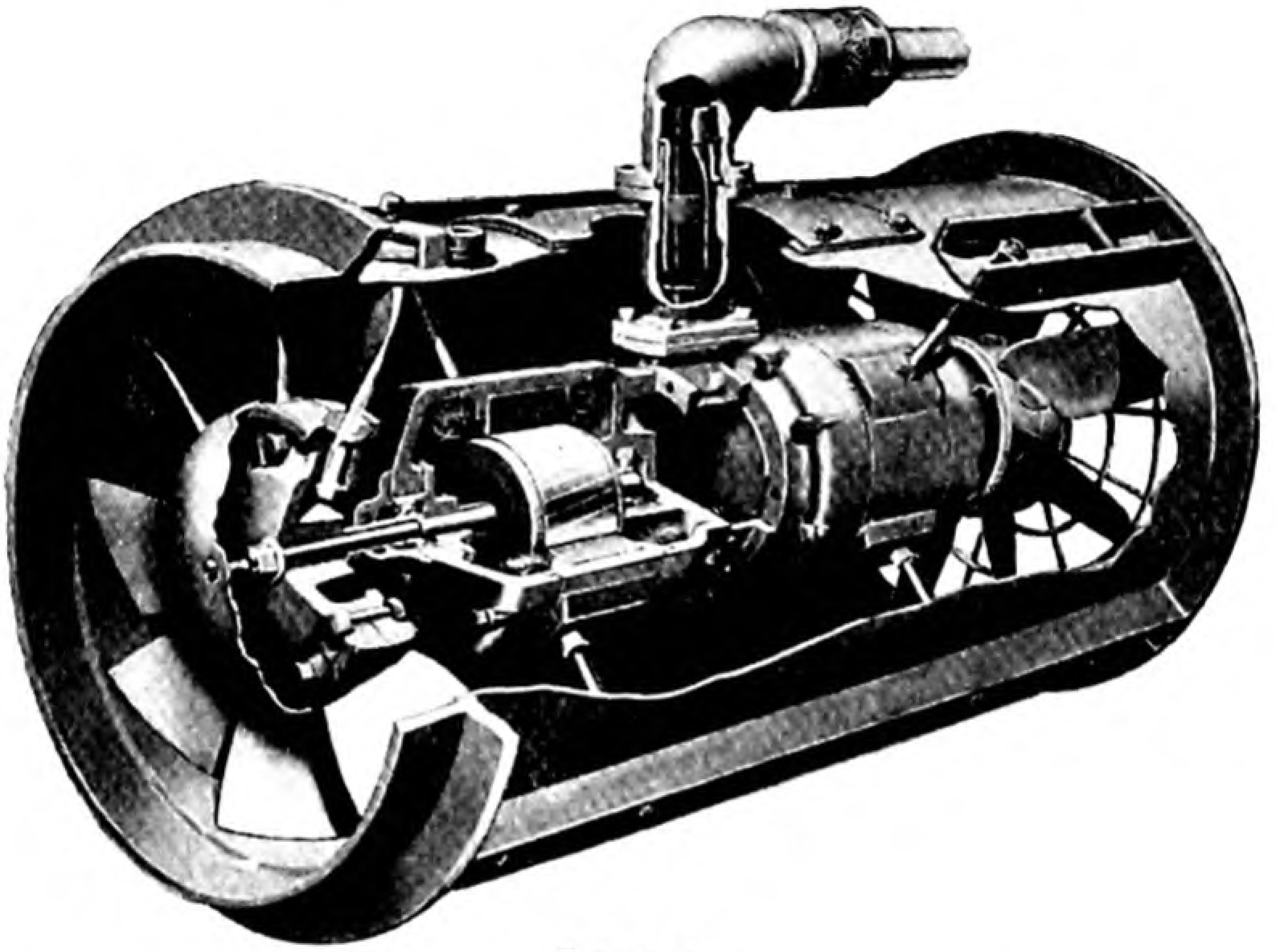


FIG. 399

by a shrunk-on steel ring. The 16-inches-diameter fan of this type is capable of delivering 5,200 cubic feet per minute of free air on an air pressure of 60 lb. per square inch.

Section 4. Fans in Series

The advantages to be gained by series installation greatly supersedes the usefulness of fans in parallel when used with ducting underground. The result of tests on two similar fans in series and in parallel is shown graphically in Fig. 401. Care has to be taken, in choosing the location of the fans in series, to avoid the creation of a 'neutral' zone in the duct line or the creation of high positive or negative pressure zones. When the fans are exhausting, the duct line should be at a negative depression throughout, and vice versa when the fans are forcing. Where leakage occurs in the duct line of an exhausting fan installation, the quantity delivered at the fan will give

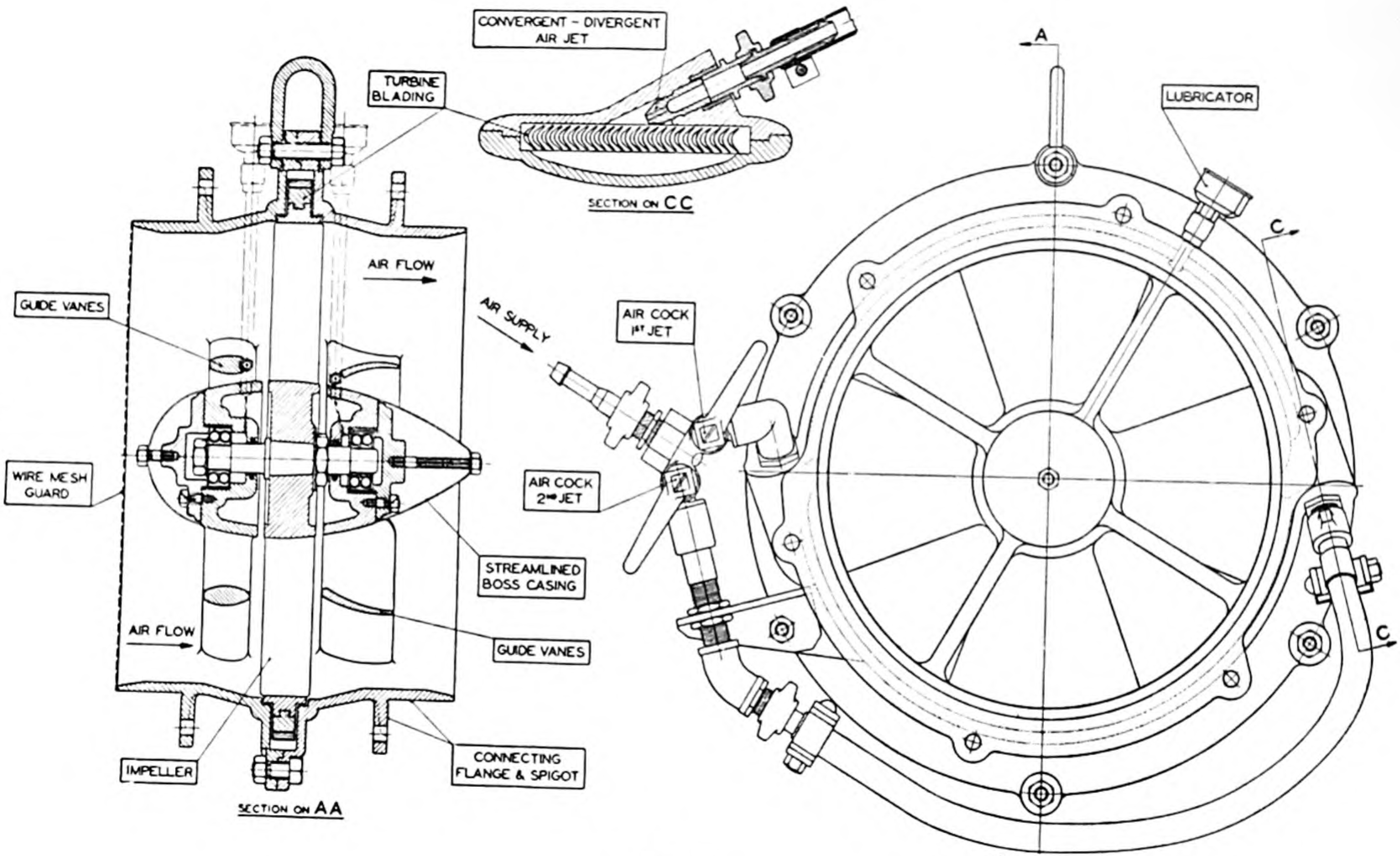

CONVERGENT - DIVERGENT AIR JET
TURBINE BLADING
SECTION ON CC
A
LUBRICATOR
C
GUIDE VANES
AIR FLOW
AIR SUPPLY
AIR COCK 1ST JET
AIR COCK 2ND JET
WIRE MESH GUARD
STREAMLINED BOSS CASING
AIR FLOW
GUIDE VANES
C
IMPELLER
CONNECTING FLANGE & SPIGOT
SECTION ON AA
A

FIG. 400

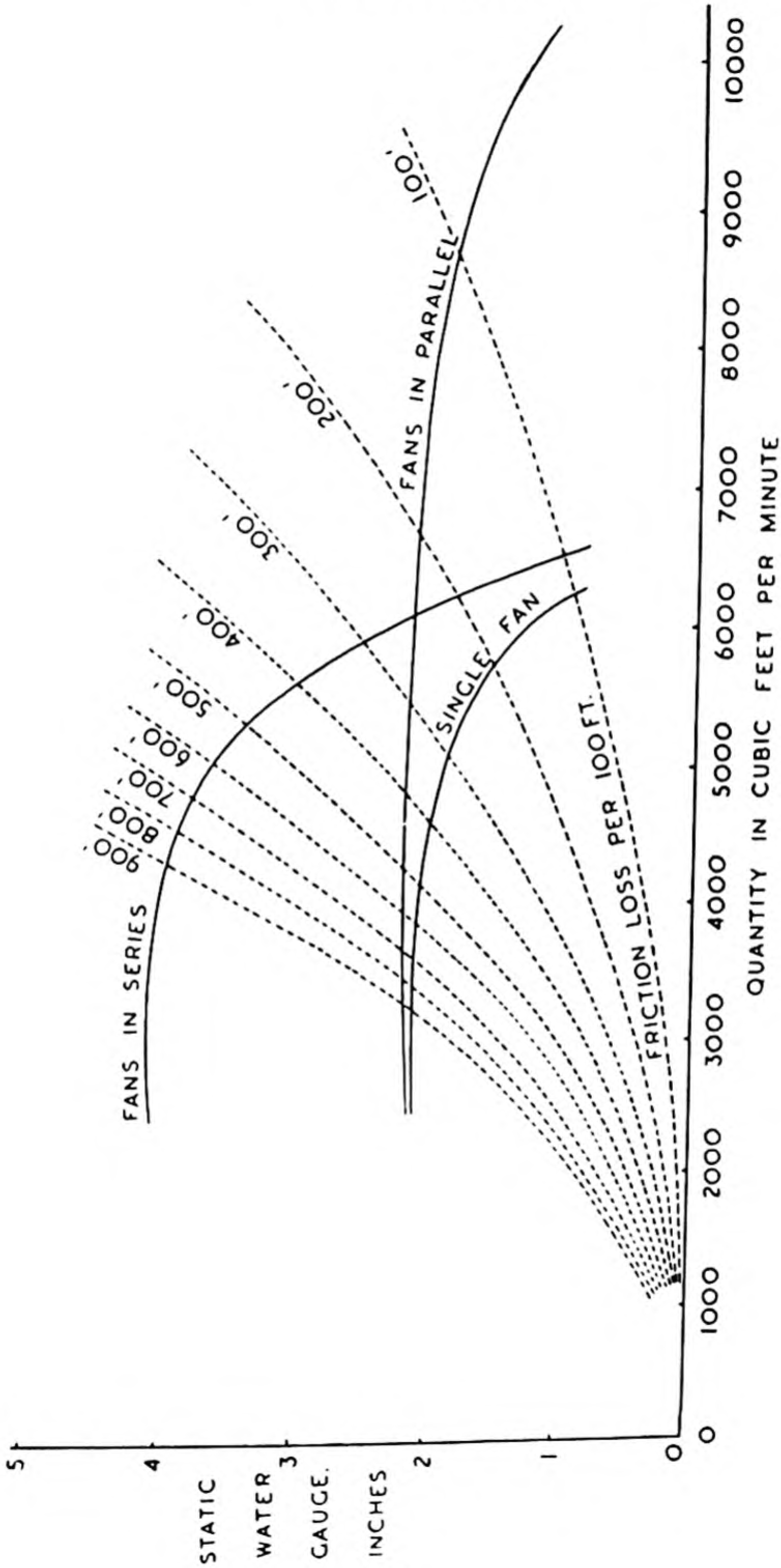

FANS IN SERIES
FANS IN PARALLEL
SINGLE FAN
FRICTION LOSS PER 100 FT.
100'
200'
300'
400'
500'
600'
700'
800'
900'
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
10000
QUANTITY IN CUBIC FEET PER MINUTE
1
2
3
4
5
STATIC WATER GAUGE. INCHES

Fig. 401

a false impression that a large volume of air is in circulation, and, in addition, the power consumption at the fan is high. Fans in series should be used only where it is impracticable to obtain the quantity required at the face with a single fan.

Where the duct line is long, the operation of one highly efficient fan is to be preferred. Auxiliary fans are now available which can provide for a varying demand at a high efficiency whether the duty is a moderate volume of air at high pressure or a large volume of air at low pressure. It is possible, therefore, to conduct auxiliary ventilation by ducting over considerable distances even with a single fan.

Section 5. Electric and Compressed-air Operation

Auxiliary fans driven by electricity or compressed air have widely different characteristics with regard to operating cost and safety, as well as with regard to their effect on the general ventilation.

The electric fan is considerably cheaper to operate than the compressed-air fan. The efficiency of the compressed-air fan is only about 30 per cent. compared to an efficiency of about 80 per cent. with electrical drive. The low degree of utilisation of the compressed air reduces the maximum value of the isothermal efficiency of compressed-air fans to about 18 per cent. compared with the mechanical efficiency of the electric fan of up to 50 per cent. A comparison of operating costs is shown in Fig. 402, in which the effect of the low efficiency of the compressed-air fan can be seen in the rising cost characteristic with increasing duct and fan diameter.

The compressed-air fan is inherently safer than the electric fan, but most electric auxiliary fans are of flame-proof design and, if installed with reasonable care, are equally safe in operation. On the Continent, the operation of electric motors in return air is often prohibited and, in consequence, only forcing ventilation is possible unless the auxiliary fan motor is situated in intake air. Where series operation is necessary, compressed-air fans are often used or, alternatively, a larger electric auxiliary fan is installed. With compressed-air drive, the cooling effect of expansion in the turbine compensates for the rise in temperature due to friction on the drive. The relative humidity of the expanded, compressed air is generally less than the air passing through the fan with which it is mixed, thus giving an additional cooling effect. In the electric fan, an increase in the air temperature is inevitable due to the form of drive.

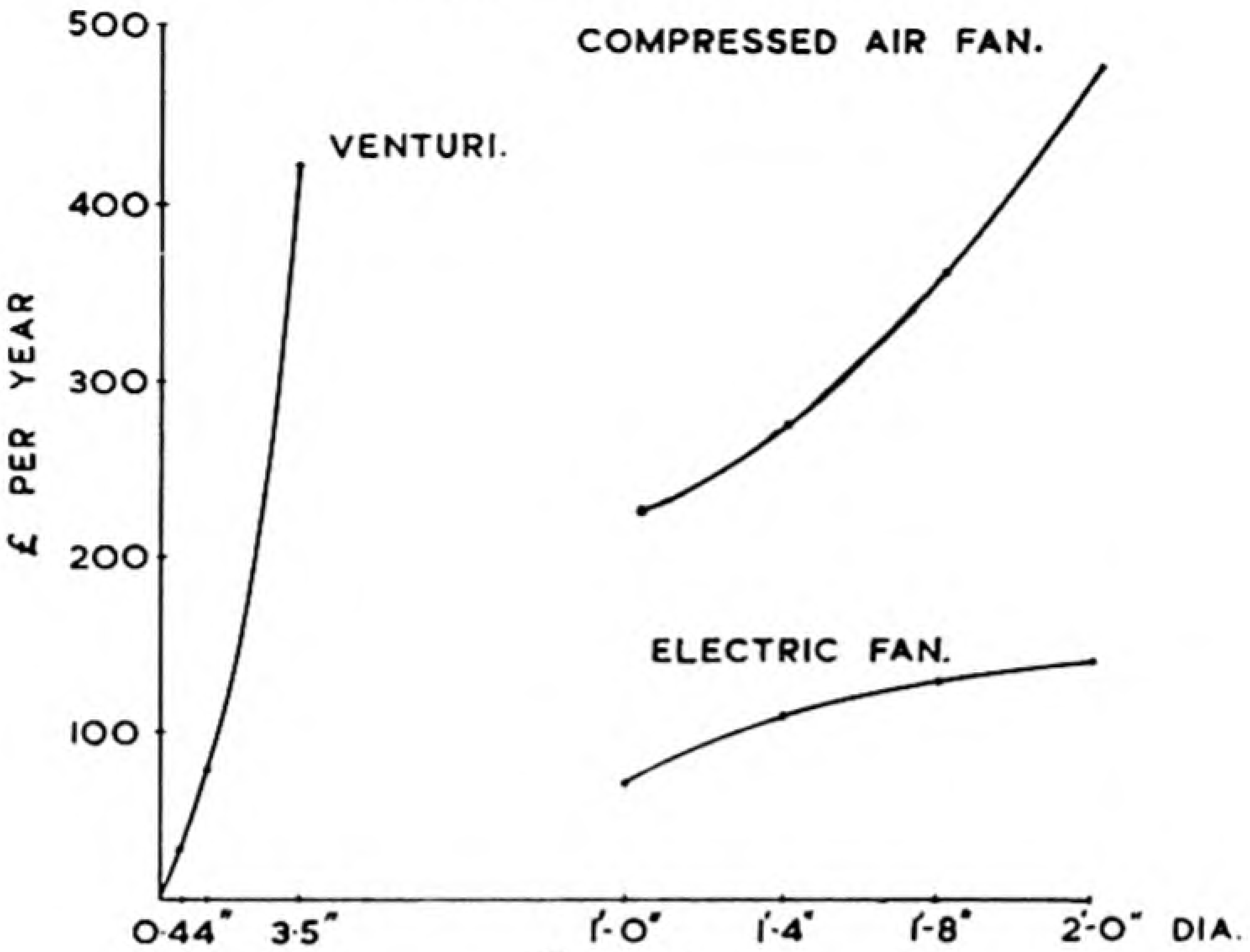


Fig. 402

Section 6. Exhausting and Forcing Ventilation

Both systems have advantages and disadvantages. In the former system the fresh air is drawn into the face via the roadway, while in the latter system the fresh air is delivered to the face through the duct line. Where it is necessary to clear shot-firing fumes or methane from the face, exhaust ventilation is often preferred to clear the face directly and as soon as possible. With exhaust ventilation, the roadway itself is kept free from these gases. Exhaust ventilation is recommended especially for rises and gate roads.

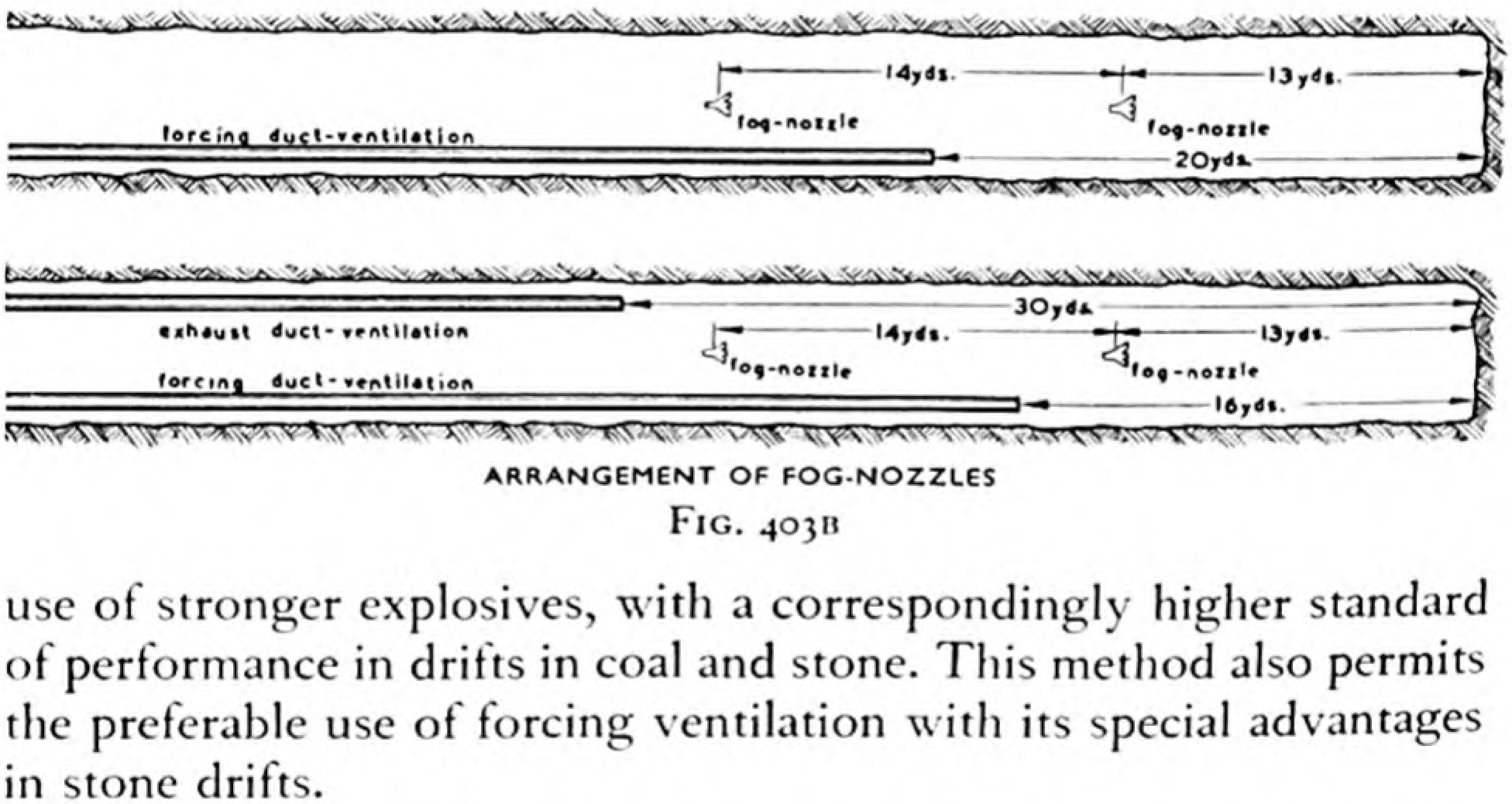
The suction effect at the end of the duct line is confined to the immediate face area near the end of the ducting, and it is recommended that a short length of smaller-diameter ducting attached to either a venturi nozzle or a small fan should be installed in front of the main duct line. This short duct line will be from about 8 to 10 yards long and is usually installed at the opposite side of the roadway and overlapping the main duct line by about 5 or 6 yards. Overlapping in this manner is absolutely essential to avoid short-circuiting or recirculation. Where an outburst of gas has to be cleared, only exhaust ventilation should be considered, since the main purpose of auxiliary ventilation is to carry away and isolate the gas as quickly as possible.

Where the atmospheric conditions at the face must be improved, forcing ventilation is preferable, since the fresh air reaches the face

in a cooler condition. To assist in keeping the intake air delivered from the duct line as cool as possible, insulated ducting may be used. Where this method is insufficient to provide the cooling effect required at the face, especially in the development of workings in hot and deep mines, refrigeration can be installed in the auxiliary ventilation system. The refrigeration unit which has been used in Continental mines is powered by a 20-h.p. motor, and has a capacity of 160,000 B.Th.U./hour. The air is passed through the unit and the cold air recirculated into the main duct line. With this system it is possible to reduce the temperature at the working face by 22° F., but it requires, in addition to the power, about 2,000 gallons per hour of cooling water. With forcing ventilation, the high exit velocity may be troublesome, and to obviate any discomfort to the men at the face a diffuser may be fitted to the end of the duct line. The diffuser can be adjusted to give lateral distribution of the air at a lower velocity, but it has the effect of absorbing part of the pressure generated by the fan, and the quantity delivered to the face is reduced. In order to obtain the advantages of both systems, reversible ventilation equipment has been used with which the normal forcing ventilation can be changed to exhausting when it is necessary to quickly clear out gas or fumes.

The main disadvantage of forcing ventilation, especially after shot firing, is removed by the recently extensively introduced fog-firing. With this method, two fog-sprays, operated by compressed air, which produce a very fine mist at the face, have been used with effect, together with forcing ventilation. Two sprays are suspended at the roof and some distance from the face as shown in Fig. 403B. The sprays have five nozzles, in which water in a finely divided state is added to the compressed-air jet. With this form of spray, illustrated in Fig. 403A, the water consumption is about 6 gallons per minute and the roadway over the whole section is filled with a fine mist or fog for a length of 25 yards. The sprays are operated for 3 minutes before and 5 minutes after firing. It has been proved by extended tests that this period is sufficient to kill the fumes produced and at the same time reduce air-borne dust by 94 per cent. The sprays have also the advantage of wetting the debris before removal, and consequently less dust is produced during the mucking operation. It has been proved by experience in the Ruhr that the introduction of mist sprays in this manner has allowed the

FIG. 403A



ARRANGEMENT OF FOG-NOZZLES

FIG. 403B

use of stronger explosives, with a correspondingly higher standard of performance in drifts in coal and stone. This method also permits the preferable use of forcing ventilation with its special advantages in stone drifts.

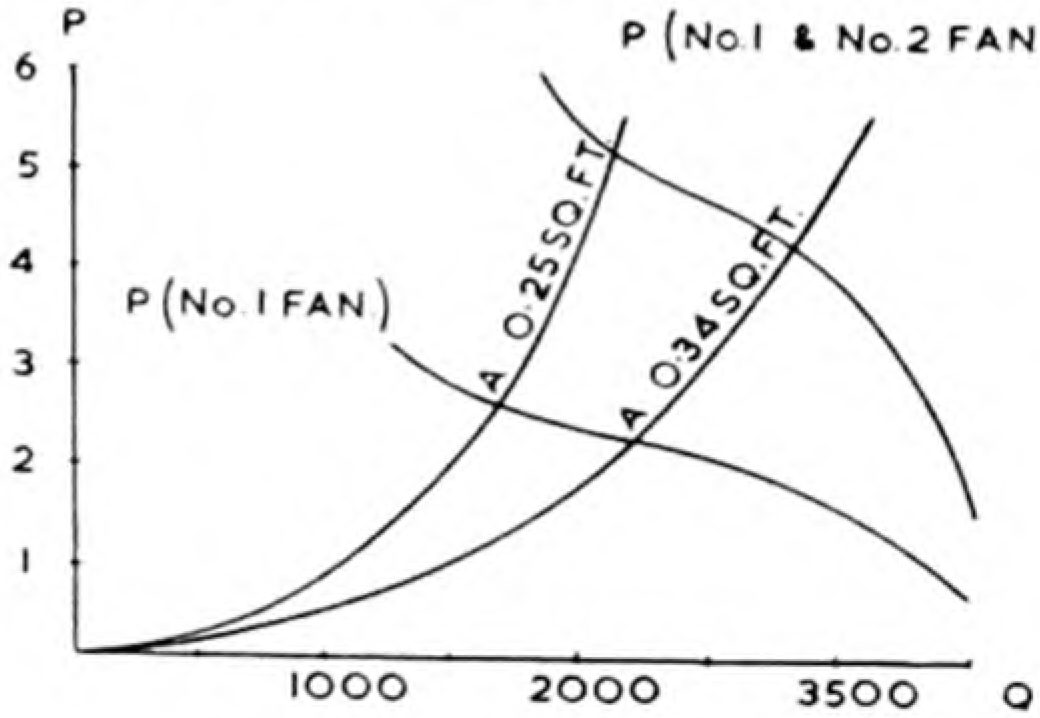
Section 7. Characteristics of Fan and Ducting—Operational Point and Efficiency

The equivalent orifice of an auxiliary ventilation duct line can be calculated in the same manner as for a mine, the area being dependent upon its length and diameter. Thus, if the equivalent orifice of the ducting is known, the corresponding ventilating pressure can be determined for a definite quantity of air, assuming a duct line free from leakage, the length of which may be called the theoretical length. If the values are plotted on a P.Q. diagram, curves are ob-

tained which are the characteristic curves for the particular ducting. The characteristic curves for two lines of air-duct are shown in Fig. 404. The larger area of orifice refers to a shorter duct line than the smaller area of orifice on the left of the diagram.

Characteristic curves for *fans* can be obtained by ascertaining the values for Q and P at the normal fan speed with changing resistances. The same figure 404 represents two such characteristic P.Q. curves for a single fan and two similar fans immediately coupled in series.

The intersection of the orifice curve and fan curves indicate the 'operational point' for the fan or fans. Thus, the single fan will deliver a quantity of 2,120 cubic feet per minute at 2·25 inches W.G.



A. EQUIVALENT ORIFICE.
Q. QUANTITY. (CUB. FT. PER MIN.)
P. STATIC PRESSURE (INCHES W.G.)

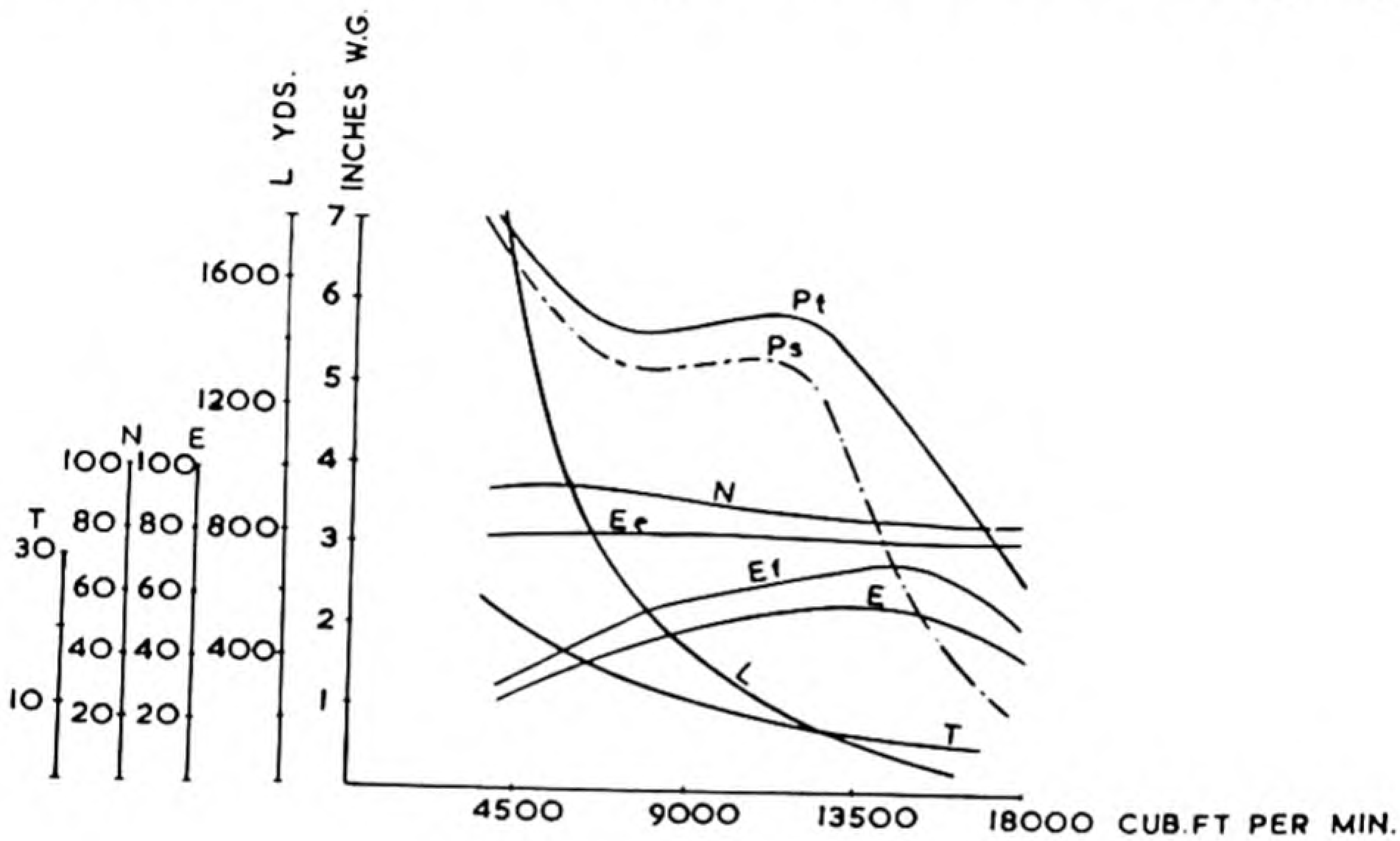
FIG. 404

in the case of the duct line with the larger orifice. When the length of ducting is increased to give the lower area of equivalent orifice, the quantity passed by the fan is reduced to 1,570 cubic feet per minute at the increased pressure of 2·7 inches W.G. Thus, an increase in the length of the ducting reduces the quantity passed by the fan, and increases the pressure at which the fan must operate. Therefore there is a limit for the length of ducting, at which the fan can operate efficiently. A fan with a steep P.Q. characteristic is more favourable than one with a flat characteristic. The characteristic for two fans in series indicates that when operating against the same equivalent orifice, the two fans will increase the quantity from 1,570 to 2,700 cubic feet per minute at twice the water gauge produced by a single fan.

The graph in Fig. 401 represents tests carried out on similar fans, singly, in series and parallel, on canvas ducting of various lengths.

The friction loss per 100 feet of ducting has been superimposed on the characteristic curves, and the benefit to be gained by the rising series characteristic is clearly demonstrated.

The curves in Fig. 405 relate to tests carried out on a special 10-h.p. electric forcing fan coupled to 2-foot-diameter ducting. The efficiency curves for the fan and motor drive, and the overall efficiency curve, are shown, together with the curve relating length



Pt. TOTAL PRESSURE
Ps. STATIC
Ee. EFFICIENCY OF ELECTRIC DRIVE.
Ef EFFICIENCY OF FAN.
E OVERALL EFFICIENCY.
L THEORETICAL LENGTH OF DUCTING.
T. AIR TEMPERATURE INCREASE. °F
N. NOISE LEVEL.

FIG. 405

of ducting, quantity and pressure, the latter increasing with the length of the line. The diagram also illustrates the rise in air temperature when forcing and the noise-level of the fan.

The total operational efficiency of the auxiliary fan installation includes the drive, fan and ducting. Since the ducting efficiency is often less than 50 per cent. in leaking ducting, an overall efficiency of 25 per cent. can be anticipated. With a good range of ducting and a well-chosen 'operational point' for the fan, the overall efficiency may be as high as 45 per cent.

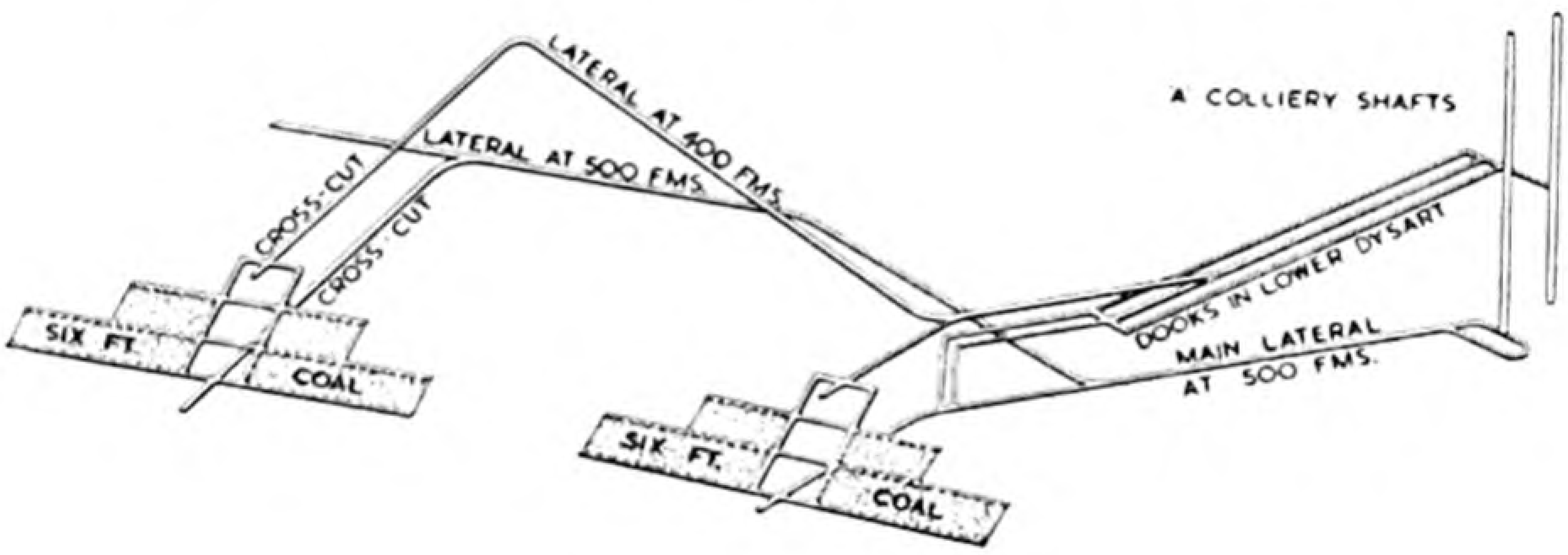
APPENDIX 1

BRITISH HORIZON MINING DEVELOPMENTS

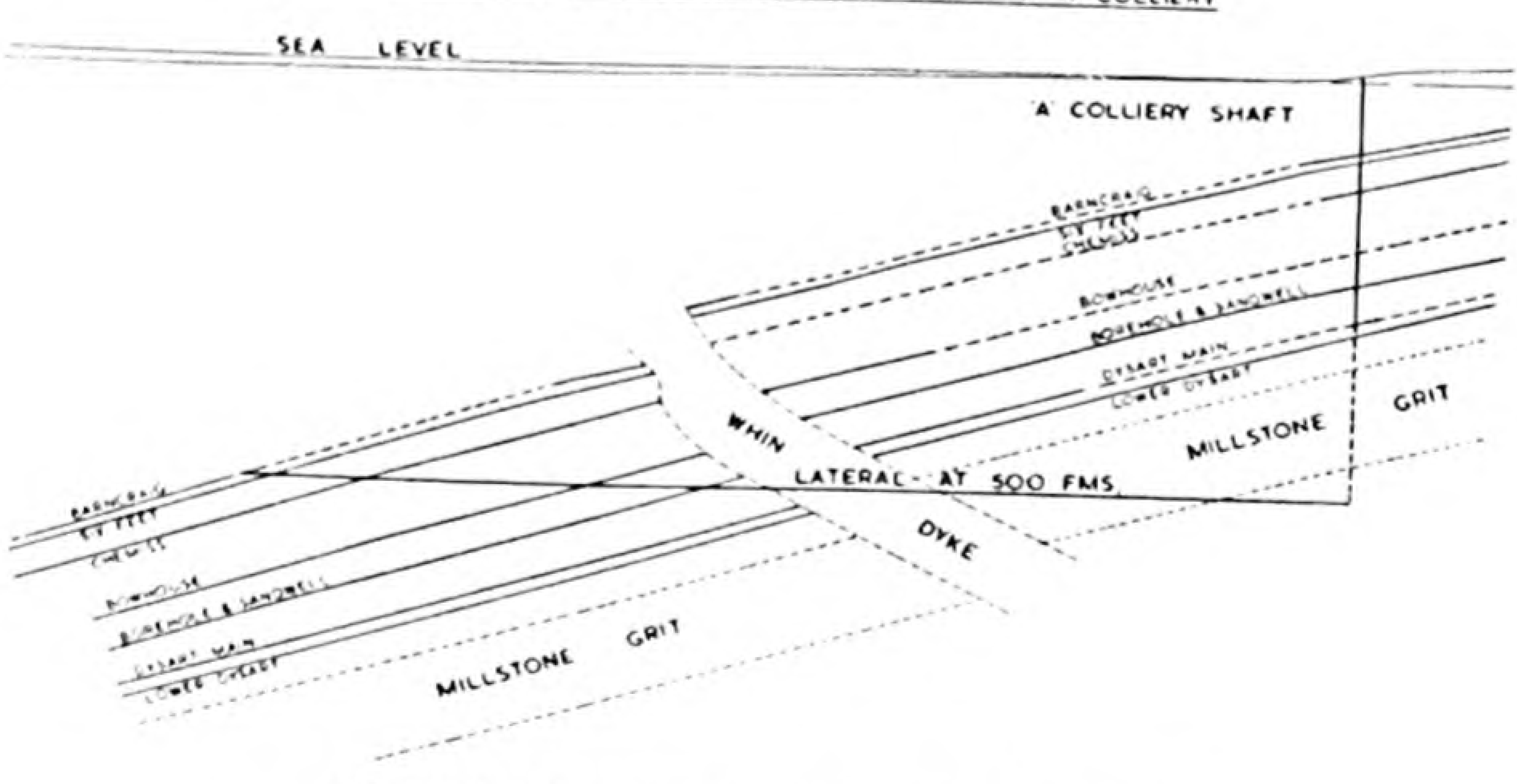
'A' Colliery

'A' COLLIERY is being reorganised to produce a capacity tonnage of 6,000 tons per day on two shifts, by a modified system of horizon mining, from a series of coal seams dipping at 1 in 2·5 to 1 in 3.

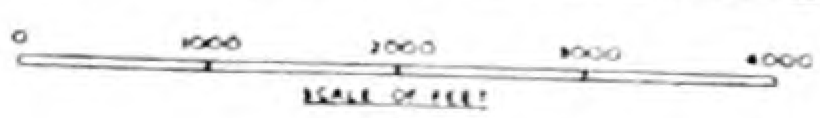
A lateral mine with cross-cuts driven in stone at 500 fathoms is to



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'A' COLLIERY



NATURAL SECTION ALONG MAIN LATERAL MINE AT 'A' COLLIERY



be used as the intake and coal haulage horizon, the returns being in the 400 fathoms horizon which is driven in stone from roadways in the Lower Dysart seam. The seams will be worked by longwall advancing methods along the strike of the seam, from dip-coal headings driven in the seams and the coal from faces between the two horizons will be conveyed down the seam roadways to loading-points on the bottom horizon.

Two shafts, each 20 feet in diameter, are available. One shaft will be equipped to have a winding capacity of 4,000 tons per day, while the other will have a capacity of 2,000 tons per day, plus men and materials winding.

The ventilation will be ascensional from the lower horizons via the seam roads to the upper horizon. One 'Aeroto' fan, plus one in reserve, giving 225,000 cu. ft. per min. at 6 in. water gauge, will be used.

The underground transport will be by 7-ton diesel locomotives at 50-h.p. and 3-ton mine cars running on a 2 feet 6 inches rail gauge from the loading-stations to the pit bottoms.

The mine is non-gassy and naked lights will be used.

The workings are expected to be wet, and to deal with this a pumping service with a capacity of 11,200 gallons per minute is to be installed.

'B' Colliery

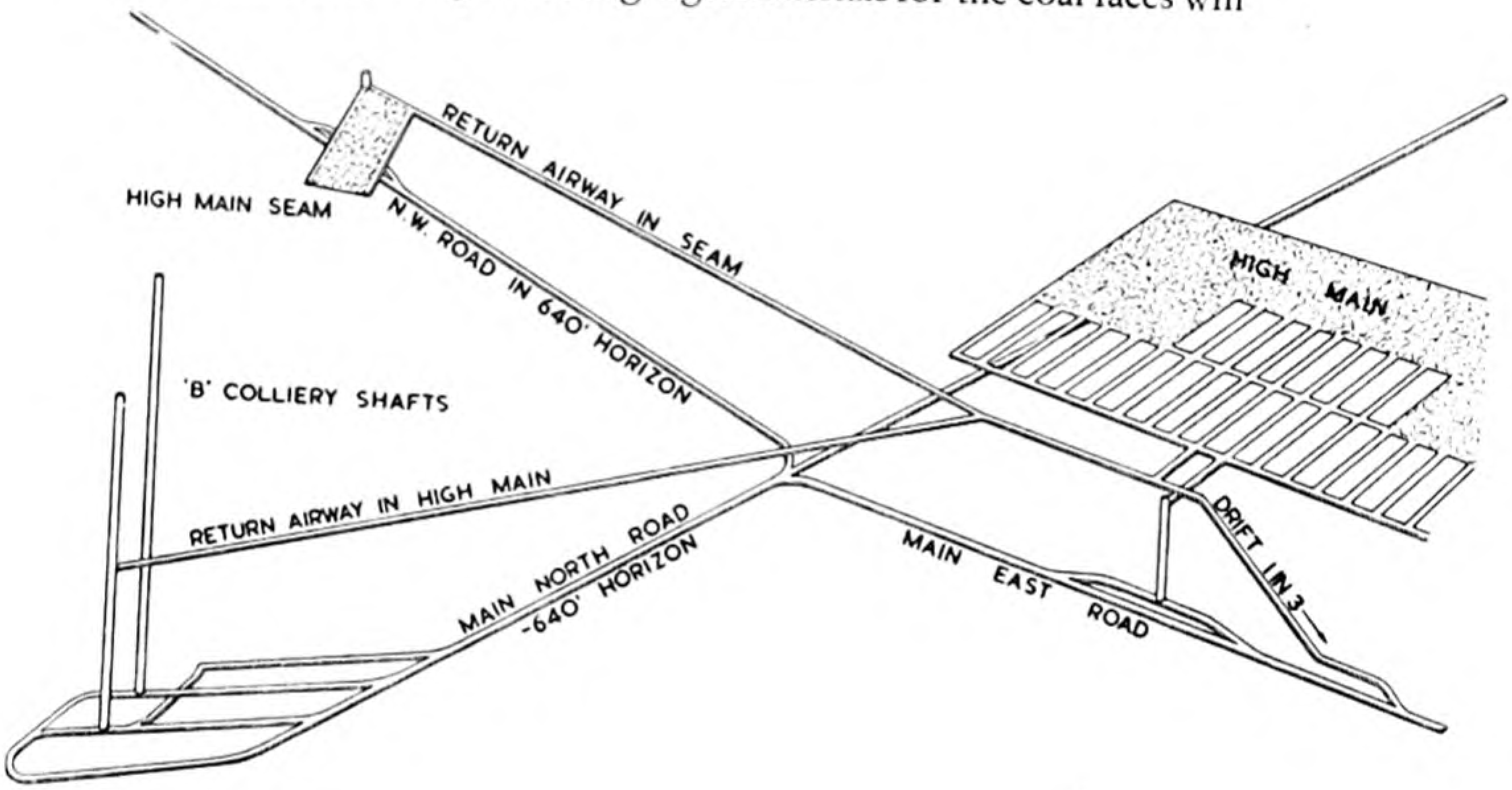
This colliery is being reorganised to produce 6,000 tons of saleable coal per day on a double-shift working by a modified system of horizon mining from a series of coal seams that dip away from the shaft at a gradient of approximately 1 in 30.

A horizon roadway at 640 feet is to be driven at 1 in 400 in the direction of full dip with laterals at both sides. These roadways are to be used as intake and coal haulage roads with return airways in the High Main seam.

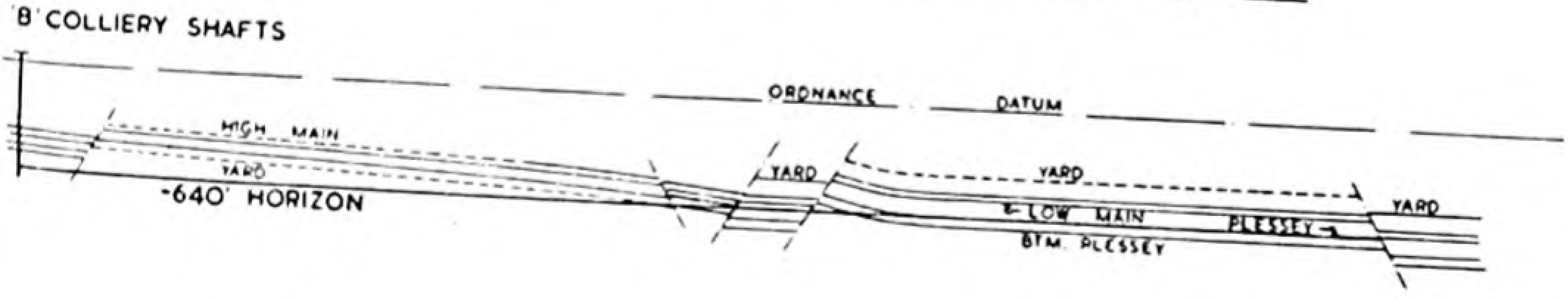
The coal will be worked partly by longwall advancing and, in some areas or seams, by bord-and-pillar partial extraction, with lines of face on the strike of the seam in both cases, and the coal being lowered down spiral chutes in staple shafts to the coal haulage road. Two shafts are available, both 18 feet in diameter with capacities of 2,500 tons per shift. Coal, men and materials will be wound at either or both shafts.

The ventilation will be ascensional from the lower horizon via staple shafts and seam roads to the return roads in the High Main seam. One Walker indestructible fan, passing 70,000 cu. ft. per min. at 1·8 inches water gauge, will be used; the seams of coal are known to be free from gas.

The underground transport will be by locomotive haulage and 5-ton mine cars on a 3-feet rail gauge. Materials for the coal faces will



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'B' COLLIERY



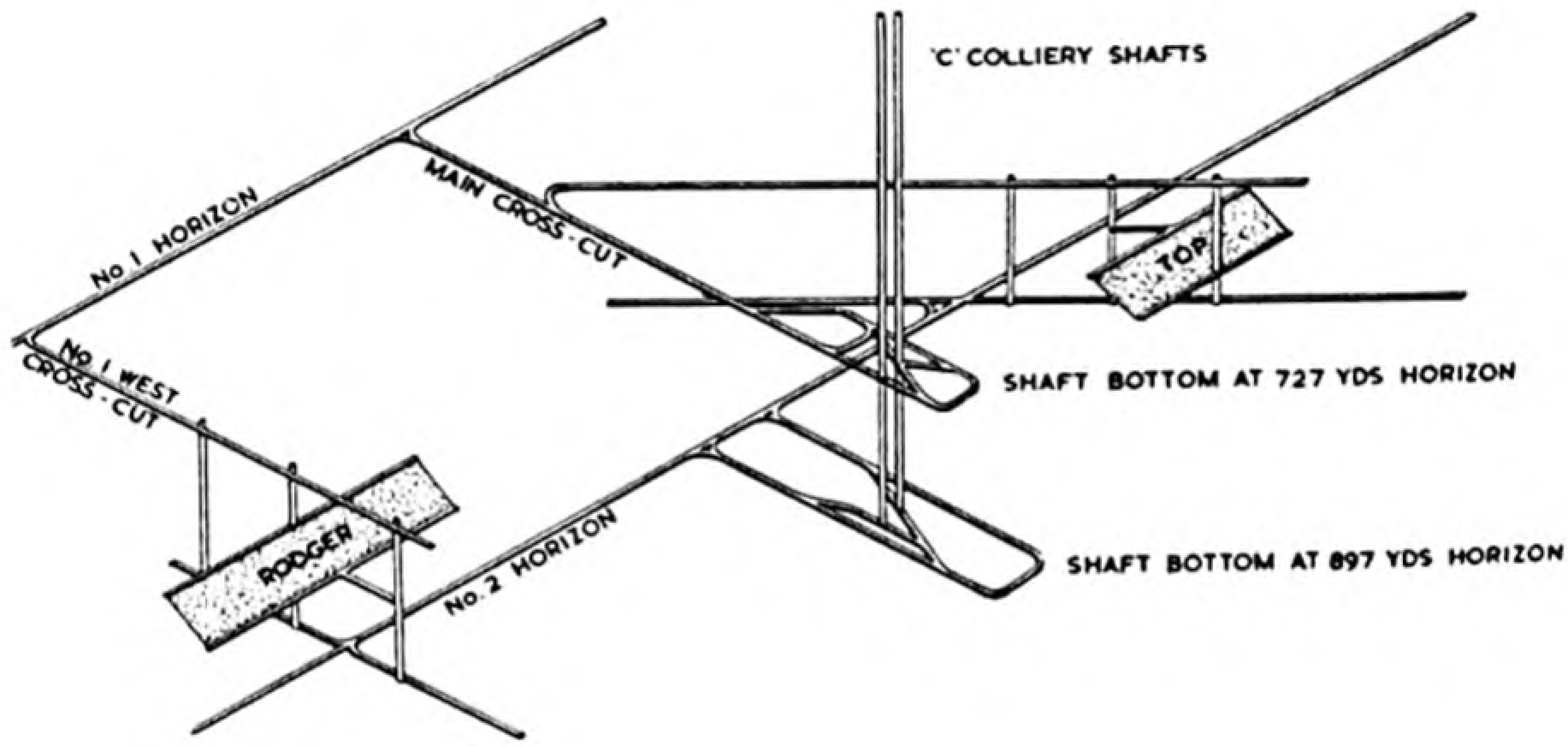
NATURAL SECTION ALONG MAIN NORTH ROAD AT 'B' COLLIERY

be taken along the coal haulage horizon, thence to the seam level up the staple pits.

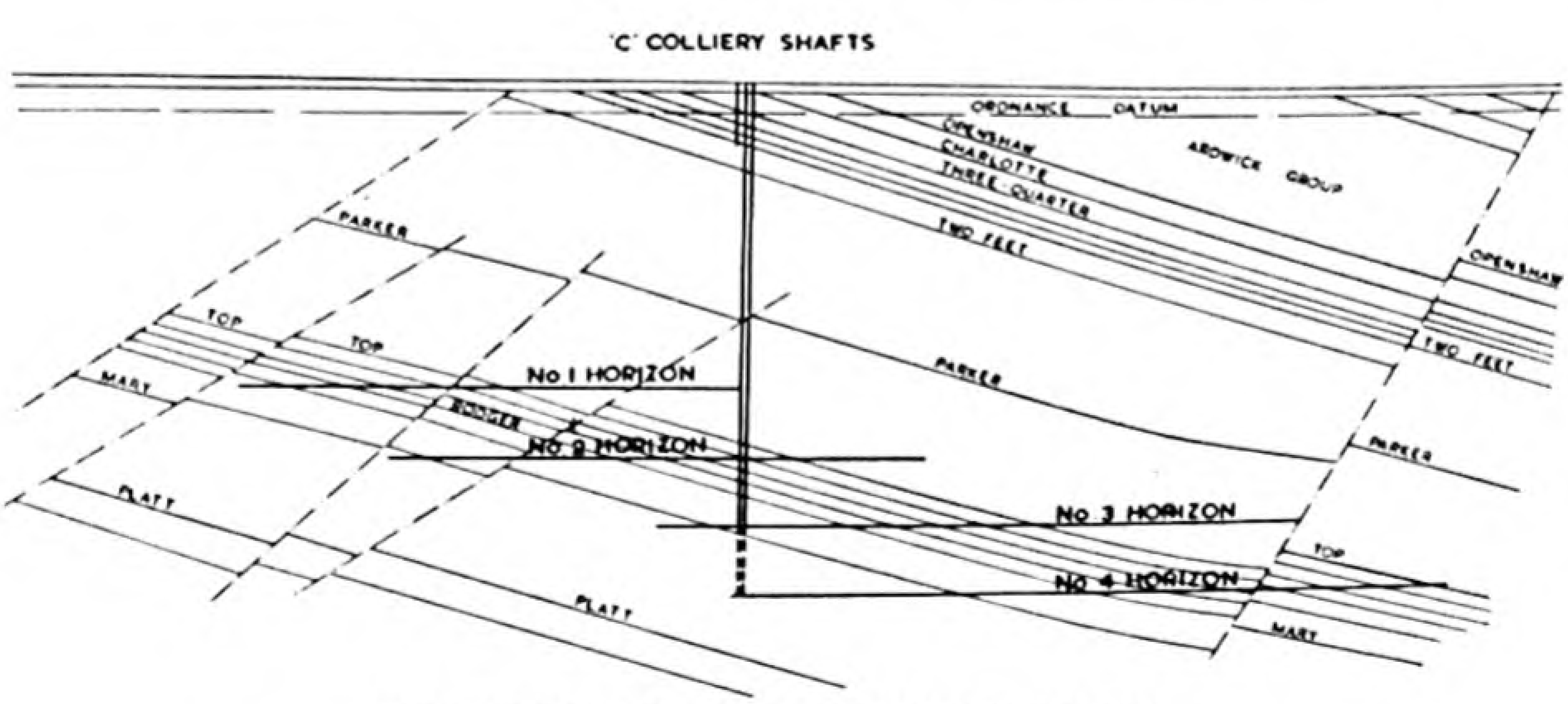
The workings are expected to be fairly wet and water lodges will be provided of adequate size to store twenty-four hour' make of water.

'C' Colliery

This colliery is being reorganised to produce 4,000 tons of coal per day, wound on two shifts, by full horizon methods of working a lower series of seams, which are badly broken by rise faults. The



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'C' COLLIERY



NATURAL SECTION ALONG MAIN CROSS-CUT AT 'C' COLLIERY

seams are steep, averaging 1 in $2\frac{1}{2}$ with some local conditions at 1 in $1\frac{1}{4}$.

Owing to the dip of the seams, four horizons are necessary: these are at depths of 727 yards from the surface, 897, 1,067 and 1,237 yards, which gives a vertical height between each of 170 yards.

The horizons will be worked in descending order with the top horizon at 727 yards being the main return for the 897 yards horizon. At each of the horizons main laterals and cross-measure drifts will be driven to work all the coal available in the take between each distinct set of horizons.

The coal will be worked by the longwall method to staple shafts from the seams to the lower horizon, the upper horizon being used for transporting men, materials and stowing dirt to the top of the staple shafts, where they will be lowered to the rise side of the working faces.

Two shafts are available, each 18 feet in diameter, one shaft being used for skip-winding at a rated capacity of 350 tons per hour, and the other for man-riding, materials and stowing dirt transport.

The ventilation will be ascensional from the lower horizon, up staple shafts to the coal seams, then by other staple shafts to the upper horizon, which will be the return roadway. An Aerex axial flow fan, electrically driven, with a capacity of 120,000 to 200,000 cu. ft. per min. at 4-in. and 8-in. water gauge, is to be erected on the surface.

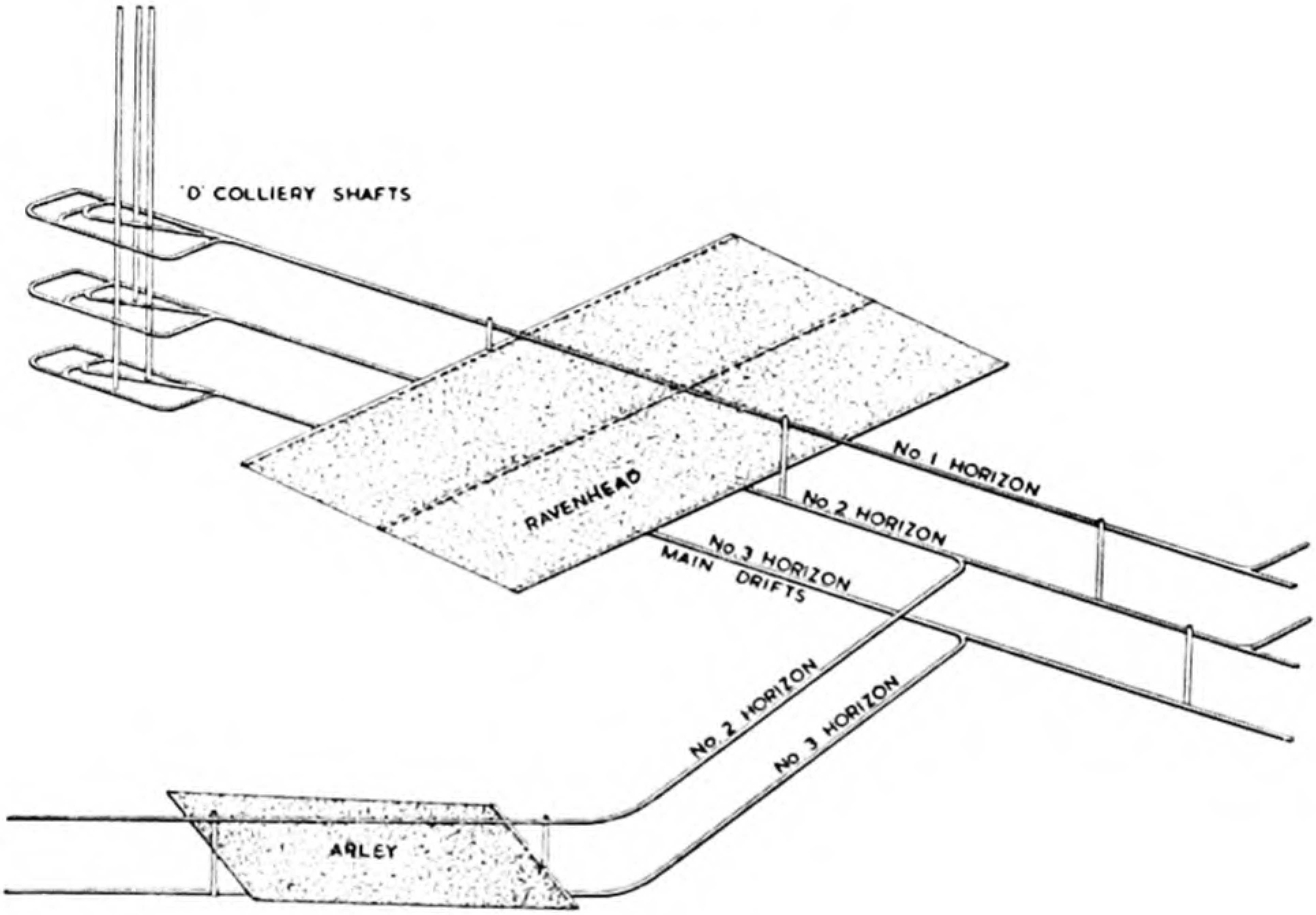
The underground transport is by locomotive haulage; battery on locomotives are to be used with 3-ton capacity mine cars on a 2 feet 6 inches rail gauge—the same cars are to be used for both coal and dirt. The staple shafts will be fitted with spiral chutes of 48 or 60 inches diameter; the size to be fixed by tests on the run of mine coal.

The seams are virtually free from water, and the gas emission is low. Pneumatic stowing of the workings is to be adopted in order to limit the effects of subsidence on the surface, which is an industrial and residential area.

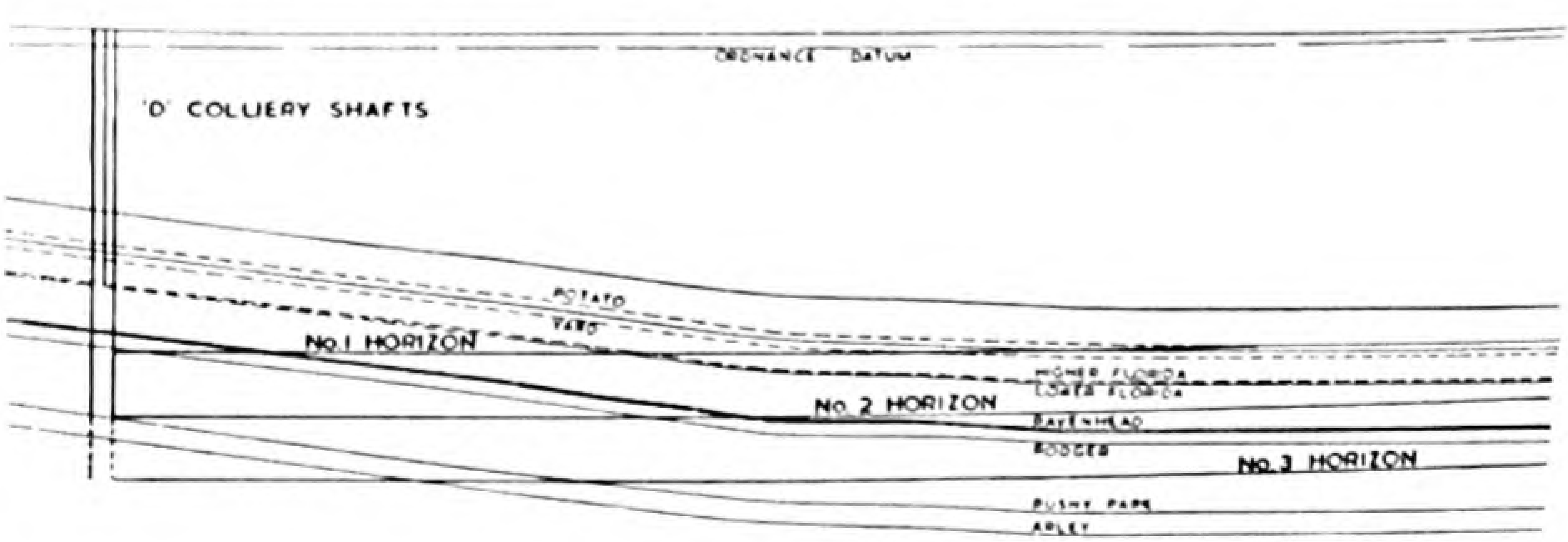
'D' Colliery

This colliery is being reorganised to produce a planned output of 6,000 tons per day, with an estimated daily saleable output of 4,700 tons of coal, wound on two shifts. The mine is planned on a modified system of horizon mining, with three horizons at 765, 915 and

1,070 yards. Each horizon will be intake haulage roads in descending order, the return airways for the top horizon will be in the Yard seam, with the top horizon the return for the other two horizons. At each of the horizons, main laterals will be driven to intersect the seams. The general dip of the seam is 1 in 7.



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'D' COLLIERY



NATURAL SECTION ALONG MAIN DRIFTS AT 'D' COLLIERY

0 1000 2000 3000 4000

SCALE OF FEET

The coal will be worked by the longwall method to staple shafts from the seams to the lower horizon, the upper horizon being used for transporting men, materials and stowing dirt to the rise-side staple shafts.

Four shafts will be available at the following diameters:

(*a*) 24 feet in diameter for coal-winding.
(*b*) 21 feet in diameter for men, materials and dirt.
(*c*) 16 feet in diameter for ventilation.
(*d*) 16 feet in diameter for ventilation pipes and cables.

The 24-feet-diameter shaft will be equipped for skip-winding, having 9-ton skips, the capacity being 600 tons per hour.

The ventilation will be ascensional from the lower horizons via staple shafts to the upper horizon, or in the case of the first horizon to roadways in the yard seam. Two axial flow fans, each of 350,000 cu. ft. per min., will be used.

The underground transport will be by diesel locomotives and 3-ton mine cars running on a 2 feet 6 inches gauge—the same cars to be used for both coal and dirt.

The seams are not excessively gassy at the neighbouring pits, but as the seams to be worked are at greater depths, provision has been made to circulate large quantities of air.

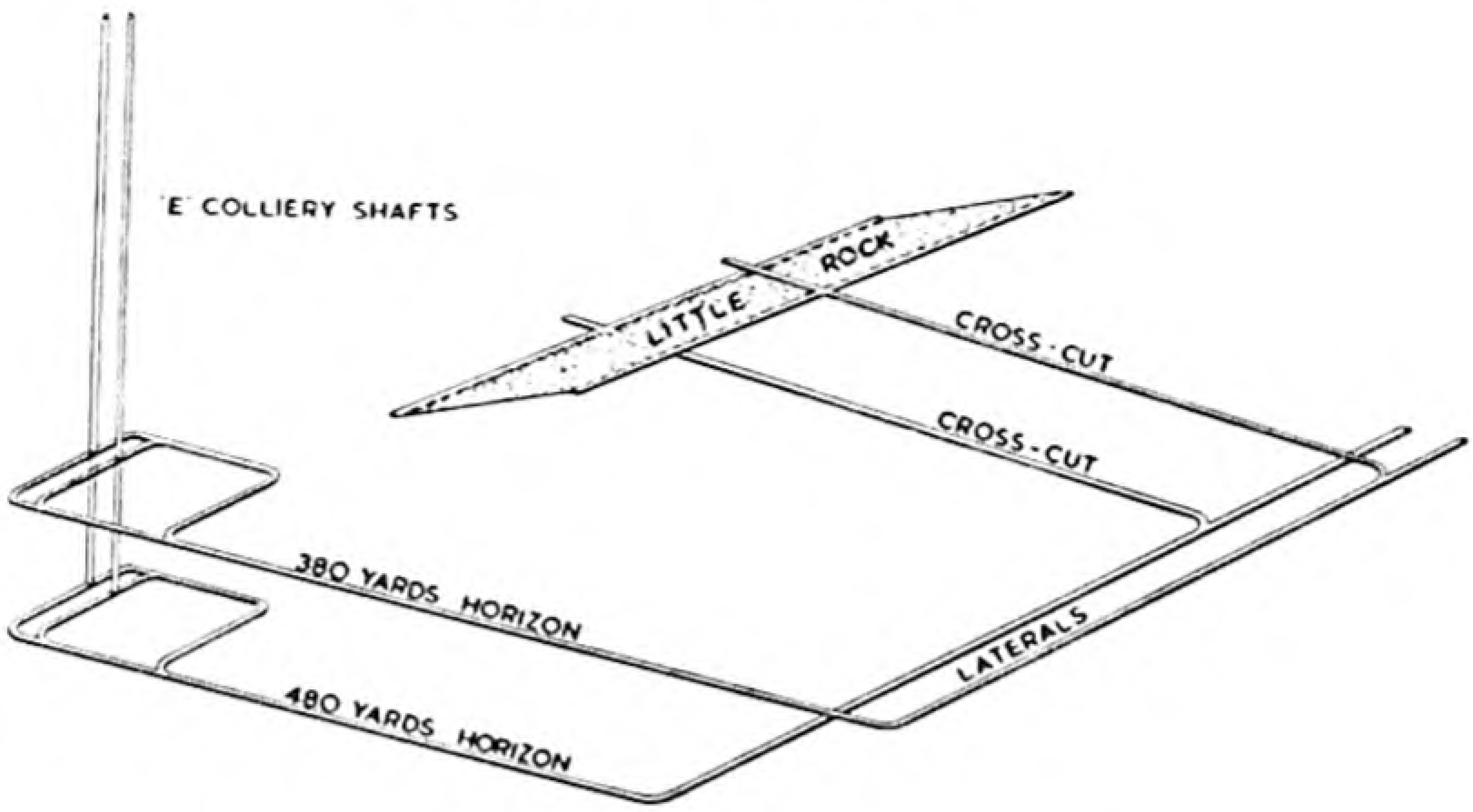
Some of the new workings will be wet and a scheme for dewatering old workings is proposed.

'E' Colliery

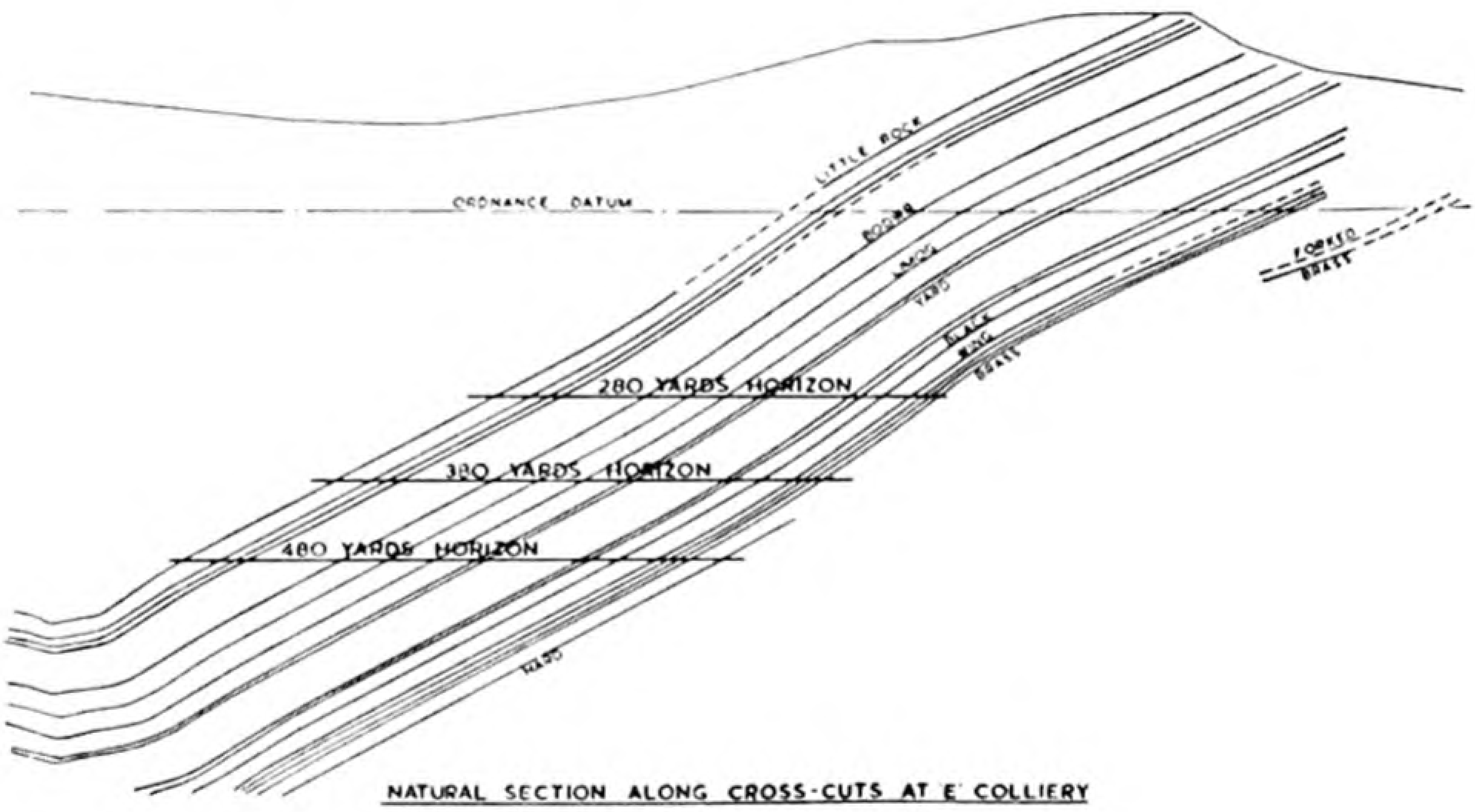
This colliery is being reorganised to work a series of sixteen coal seams dipping at 1 in 3·3 to 1 in 7·5 by the horizon method of mining. The planned saleable output is 1,250 tons per day on two shifts.

A horizon system with laterals and cross-cuts at 480 yards from the surface is to be driven. These roadways are to be the intake airways and coal-hauling roads, and the return airways will be in the 380 yards horizon roadways which are to be driven for this purpose. Each seam will be worked by the longwall advancing method, the gate roads being on the strike of the seams. The seams will be worked on one face between the horizons.

Three shafts will be available, one 13 feet in diameter for coal-winding, a second at 6 feet diameter for ventilation, with a third at



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'E' COLLIERY



NATURAL SECTION ALONG CROSS-CUTS AT 'E' COLLIERY

0 1000 2000 3000 4000

SCALE OF FEET

4 feet 6 inches diameter for emergency ventilation. The first shaft, with a capacity of 125 tons per hour, will wind coal, men and materials.

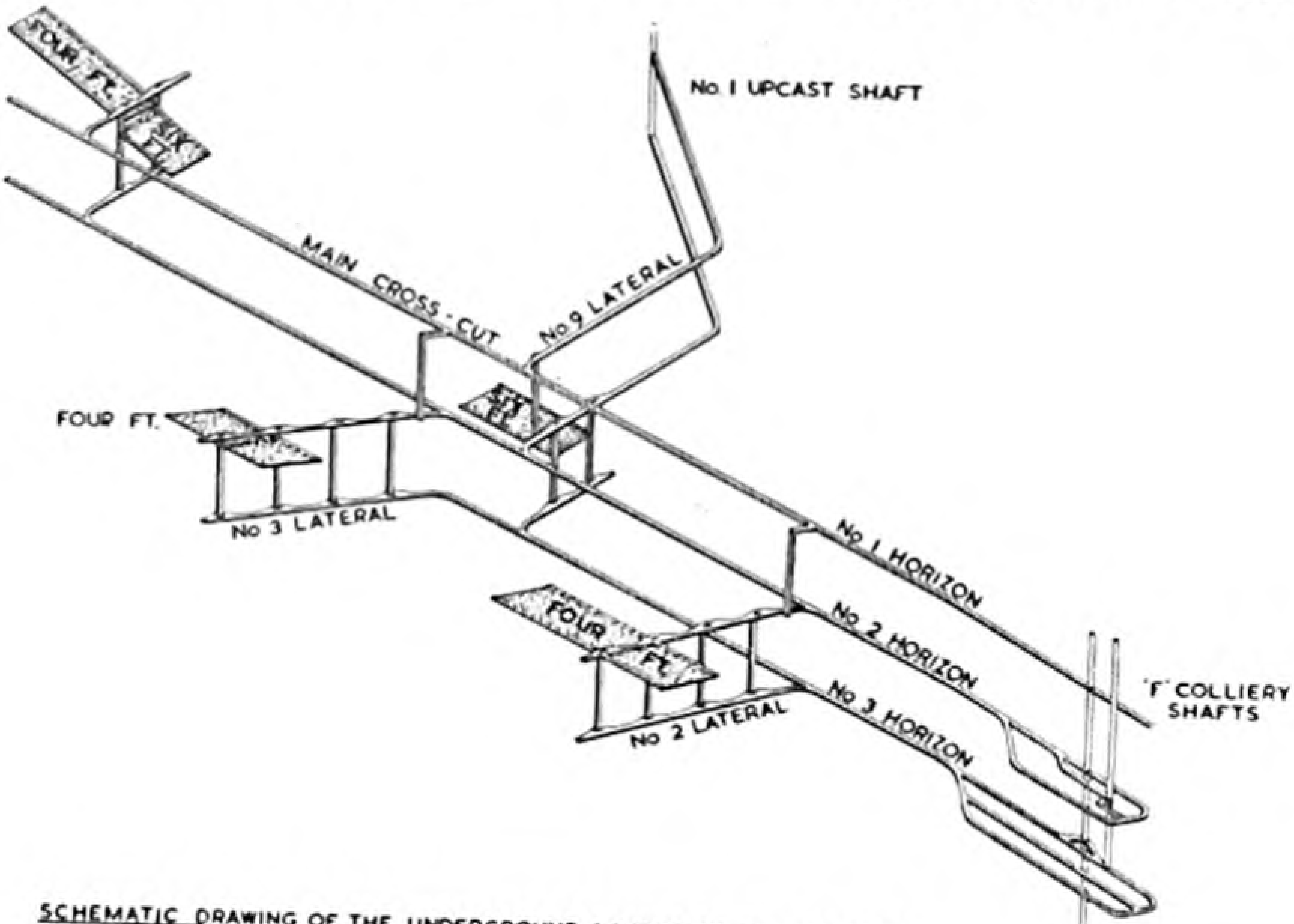
The ventilation will be ascensional from the lower horizon via seam roads to the upper horizon. One Keith-Blackman fan at 33,000 cu. ft. per min. and 4 inches water gauge will be used.

The underground transport is to be by 10-ton battery locomotives and 14-cwt. tubs on a 2 feet $5\frac{1}{2}$ inches rail gauge from loading-points to the pit bottom.

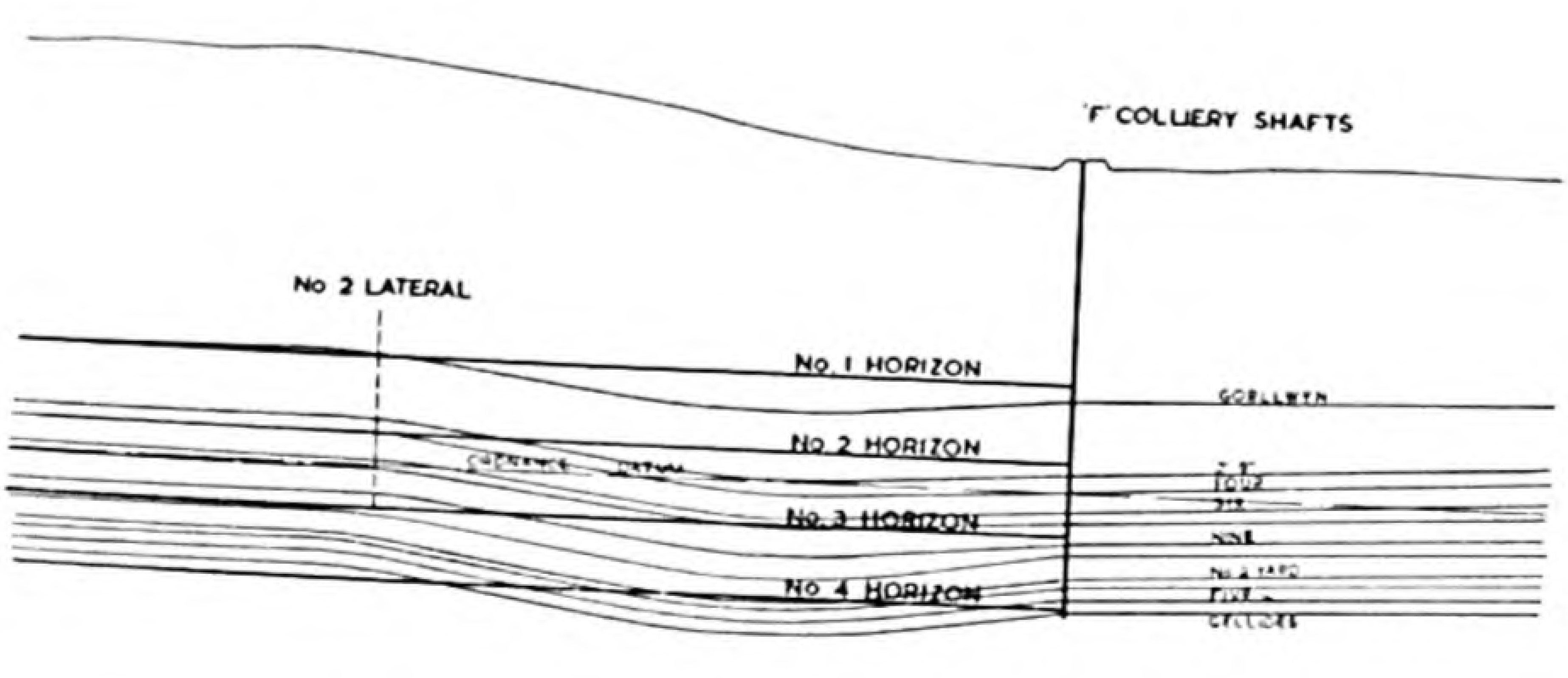
The mine is rated non-gassy, and the make of water is approximately 10,000 gallons per hour.

'F' Colliery

This colliery is being reorganised to produce a planned saleable output of 4,800 tons per day on a double winding shift, from a series



SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'F' COLLIERY



NATURAL SECTION ALONG MAIN CROSS CUT AT 'F' COLLIERY

of coal seams that are slightly undulating, with an average dip of 1 in 20.

Full horizon mining is to be adopted with horizons at 242, 305, 408 and 485 yards. The 305 and 408 yards horizons are to be used simultaneously as intake and coal haulage roadways, with the 242 yards horizon being the common return for both Nos. 2 and 3.

The seams are to be worked by single-unit longwall faces 200 yards long, advancing on the strike of the seam either to the rise or to the dip, as necessary.

Three shafts are available as follows:

No. 1. 18 by 13 feet (elliptical), to be used for man-riding.

No. 3. 16 feet in diameter, with a capacity of 2,500 tons per day on double shift, to be used for winding coal, dirt, materials and men.

No. 4. 16 feet in diameter, with a capacity of 3,500 tons per day on a double shift, to be used also for coal, dirt, men and materials.

The ventilation will be ascensional from the lower horizons via staple shafts and seam roadways to the top horizon. Two fans, passing 335,000 cu. ft. per min. at 13·5 inches water gauge, are to be used: this has been found necessary, as the seams are known to be gassy.

Underground haulage will be by belt conveyors to spiral chutes in the staple shafts, loading at the bottom into 3-ton mine cars on a 3-foot rail gauge, the mine cars being hauled to the pit bottom by 10-ton, 65-h.p., diesel locomotives.

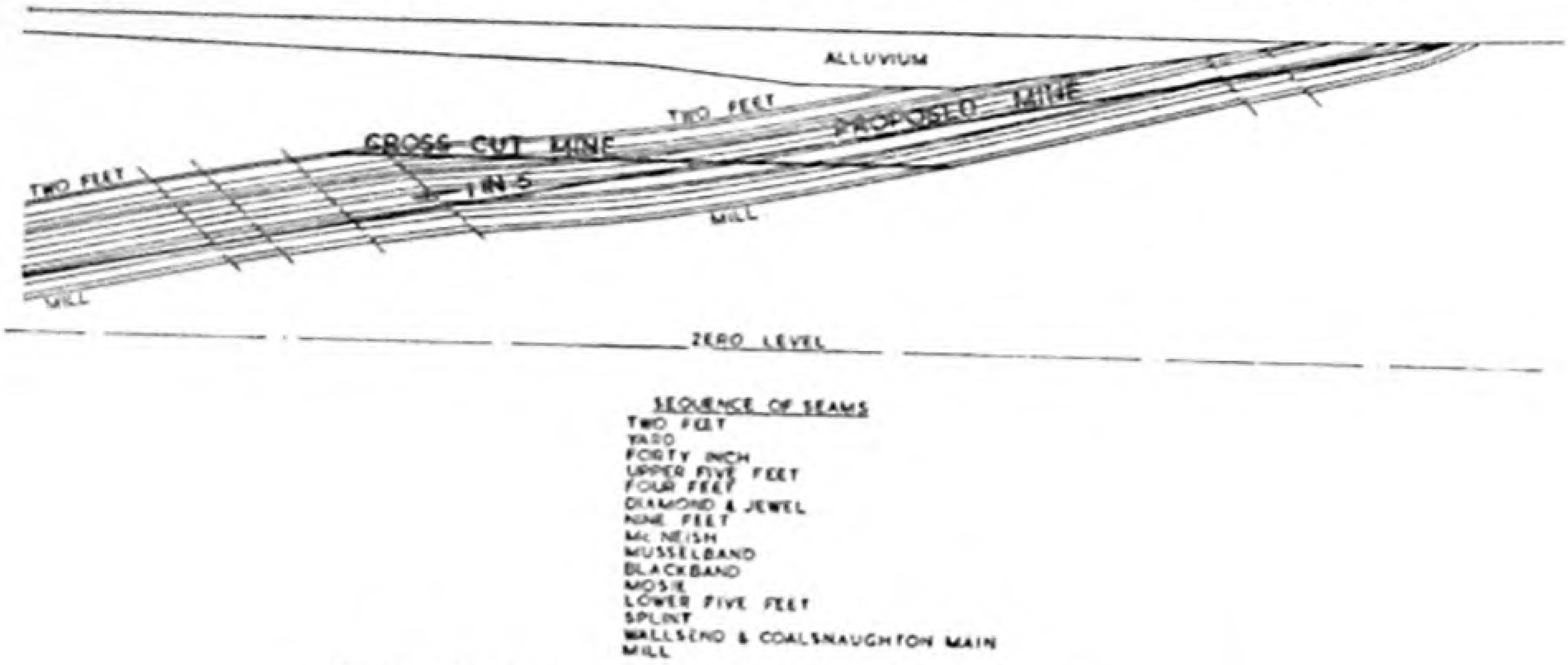
The faces will be pneumatically stowed, the stone for which will be transported along the horizon that does not serve the individual face in a coal-haulage capacity, i.e. No. 1 horizon will serve faces on No. 2 horizon; likewise No. 2 horizon will serve faces on No. 3 horizon, etc.

It is not expected that there will be large quantities of water to be pumped, as the seams are known to be fairly dry.

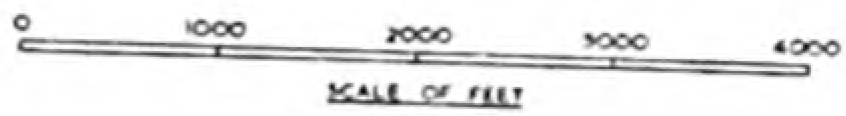
'G' Colliery

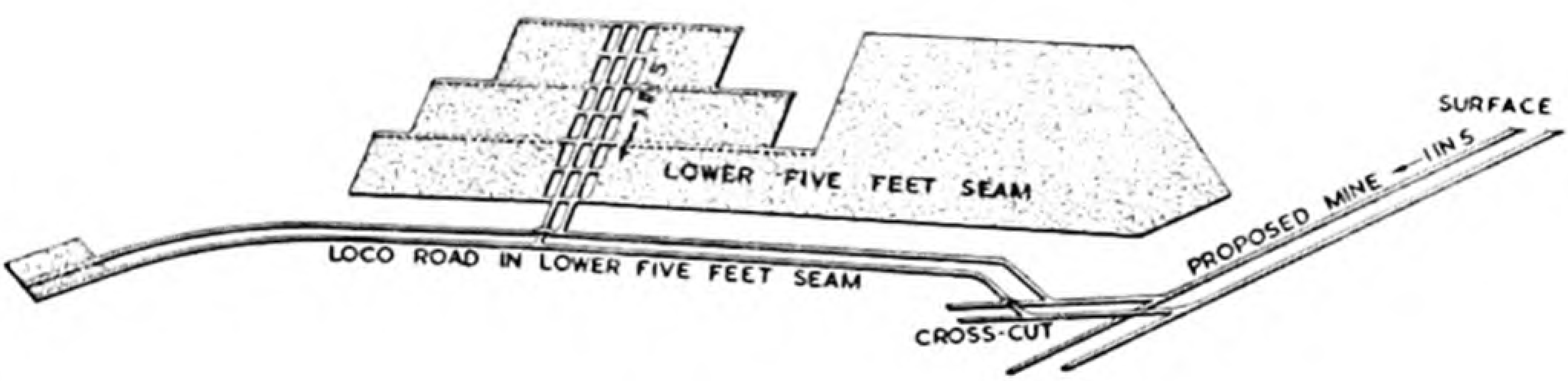
This colliery is a new surface-mine project which is planned to produce 3,000 tons per day on two shifts.

The area to be worked lies in a basin, the depth to the lowest seam at the bottom of the basin being approximately 600 yards and the average gradient of the seams being 1 in $4\frac{1}{2}$. There are sixteen



NATURAL SECTION ON LINE OF PROPOSED MINE AT 'G' COLLIERY





SCHEMATIC DRAWING OF THE UNDERGROUND LAYOUT AT 'G' COLLIERY

seams to be worked, varying in thickness from 1 foot 6 inches to 6 feet 3 inches. A proportion of this coal, especially in the thicker seams on the edges of the basin, have already been worked, mostly by room-and-pillar method of working. These workings are at present waterlogged.

To win the coal, it is proposed to drive two parallel stone drifts 15 feet wide by 11 feet high at a gradient of 1 in 5 from the surface to intersect the seams. At a distance of 1,500 yards from the drift mouth, a level cross-cut intersects the seams. From this cross-cut, locomotive roadways will be driven right round the basin following the level course in the seams. From this locomotive roadway, coal headings will be driven up the full dip of the seam to form single-

unit longwall advancing faces to the right and left with conveyor and tailgates on the strike of the seam.

The underground method of transport will be by belt conveyors from the face to a loading-point at the bottom of the main rise heading on the level locomotive seam road. At this point the coal will be loaded into 3-ton mine cars on 2 feet 6 inches rail gauge, these will be hauled by locomotives to the bottom of the drift. The coal will be transferred at this point into 9-ton 'Monitor' drop-bottom mine cars and then by main rope haulage to the surface.

The water at this colliery will be dealt with by a drainage scheme which is being negotiated for the areas.

'H' Colliery

This is a colliery which is to be reorganised to produce a planned saleable output of 3,360 tons of coal per day on a double-shift working from a series of coal seams with an average dip of 1 in 25.

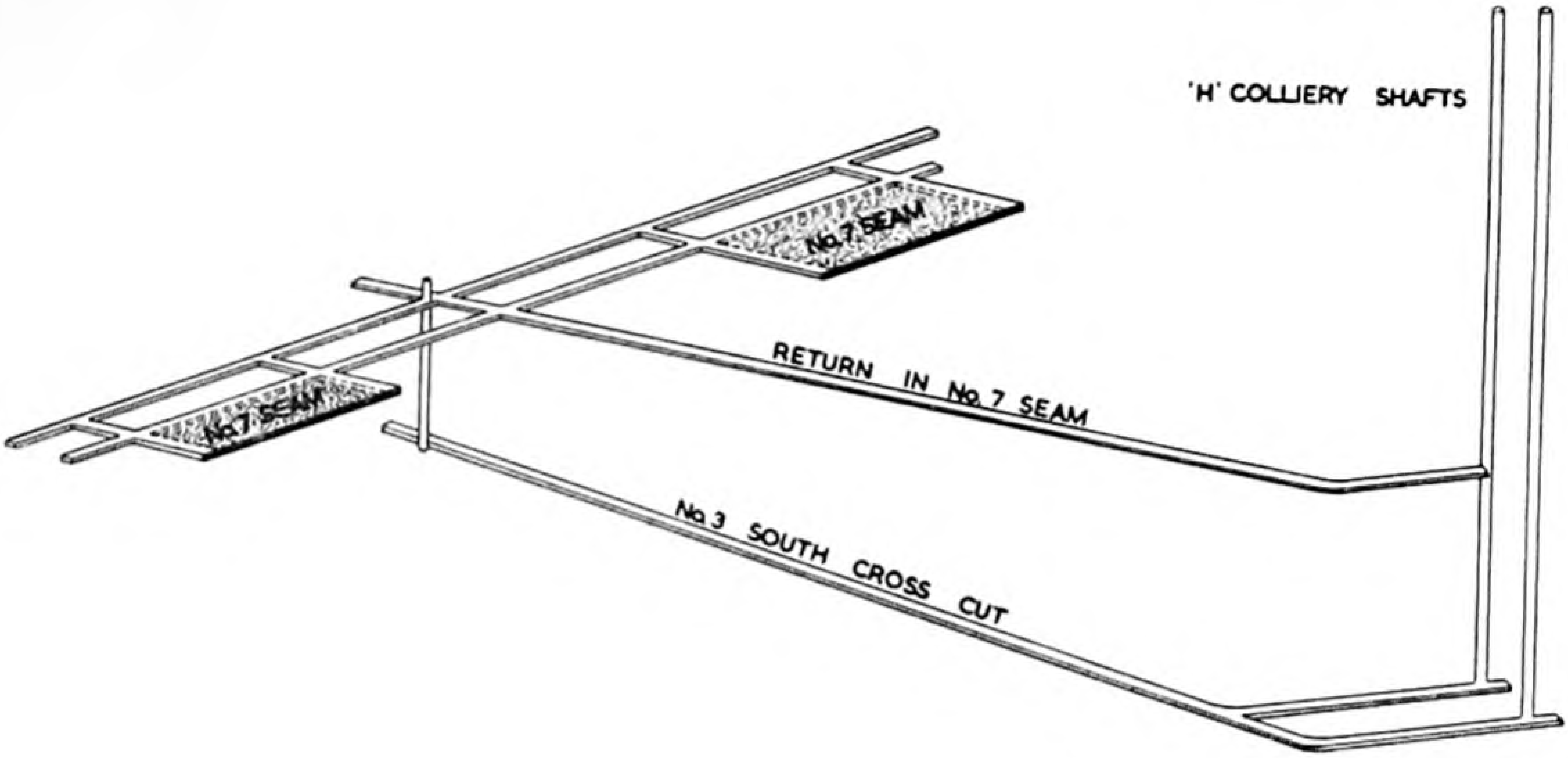
The method to be adopted is a modified system of horizon mining. The shafts are to be deepened to the base of the coal measures and a horizon driven at 1,700-feet to intersect the seams. The seams will be worked by longwall advancing, from staple shafts driven from the 1,700-feet horizon roads to roadways that are existing in the top seam which has been fairly extensively worked.

Two shafts, each 16 feet in diameter, will be made available, both having a capacity of 1,800 tons per shift; one shaft will be used for coal-winding, the other for winding men and materials and overflow coal from the other shaft during peak periods.

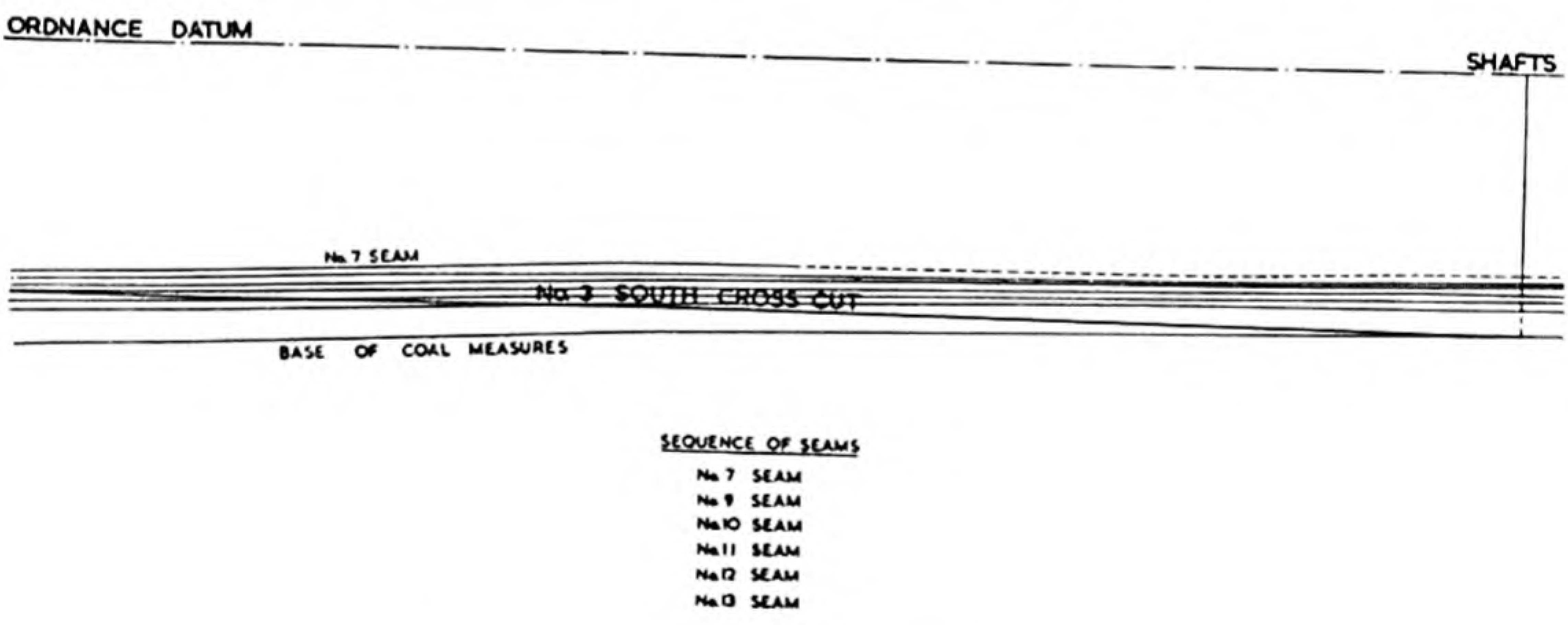
The ventilation will be ascensional from the lower horizon via staple shafts and the seam roadways, to existing return roadways in the top seam. One 'Aeroto' fan, passing at 400,000 cu. ft. per min. at 5 inches water gauge, is to be used.

The underground transport will be by 12-ton, 80-h.p., diesel locomotives and $2\frac{1}{2}$-ton mine cars on a 2-foot rail gauge from the loading-station at the bottom of the staple shafts, which will be equipped with spiral chutes, to the pit bottom.

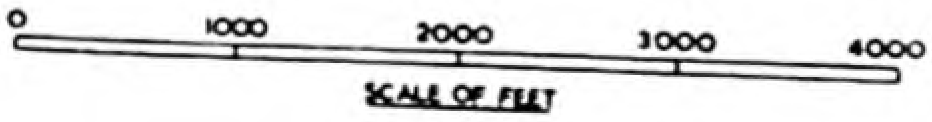
In the seams to be worked gas emission is known to be practically nil.



SCHEMATIC DRAWING OF UNDERGROUND LAYOUT AT 'H' COLLIERY

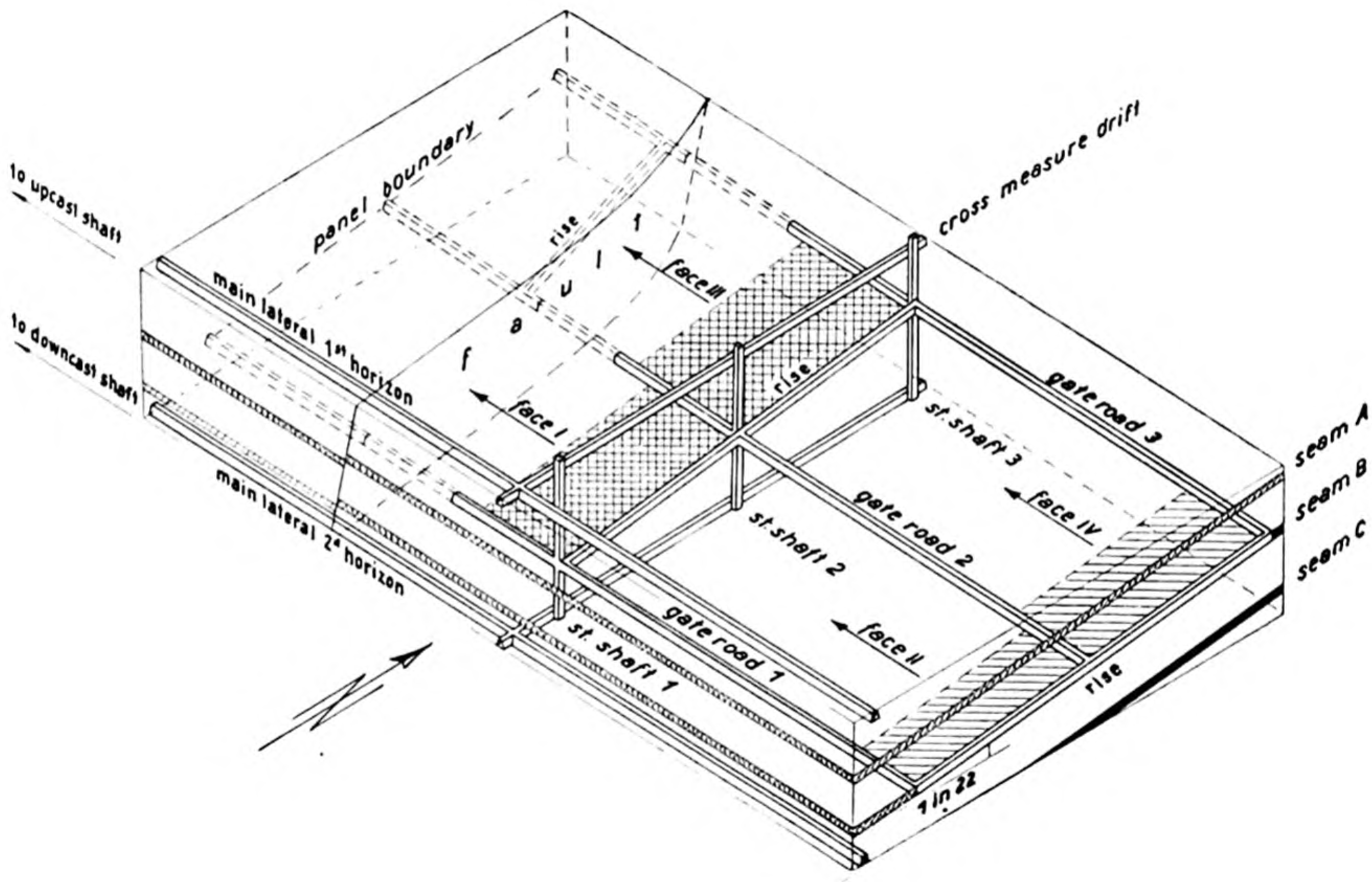


NATURAL SECTION ALONG No. 3 SOUTH CROSS CUT AT 'H' COLLIERY

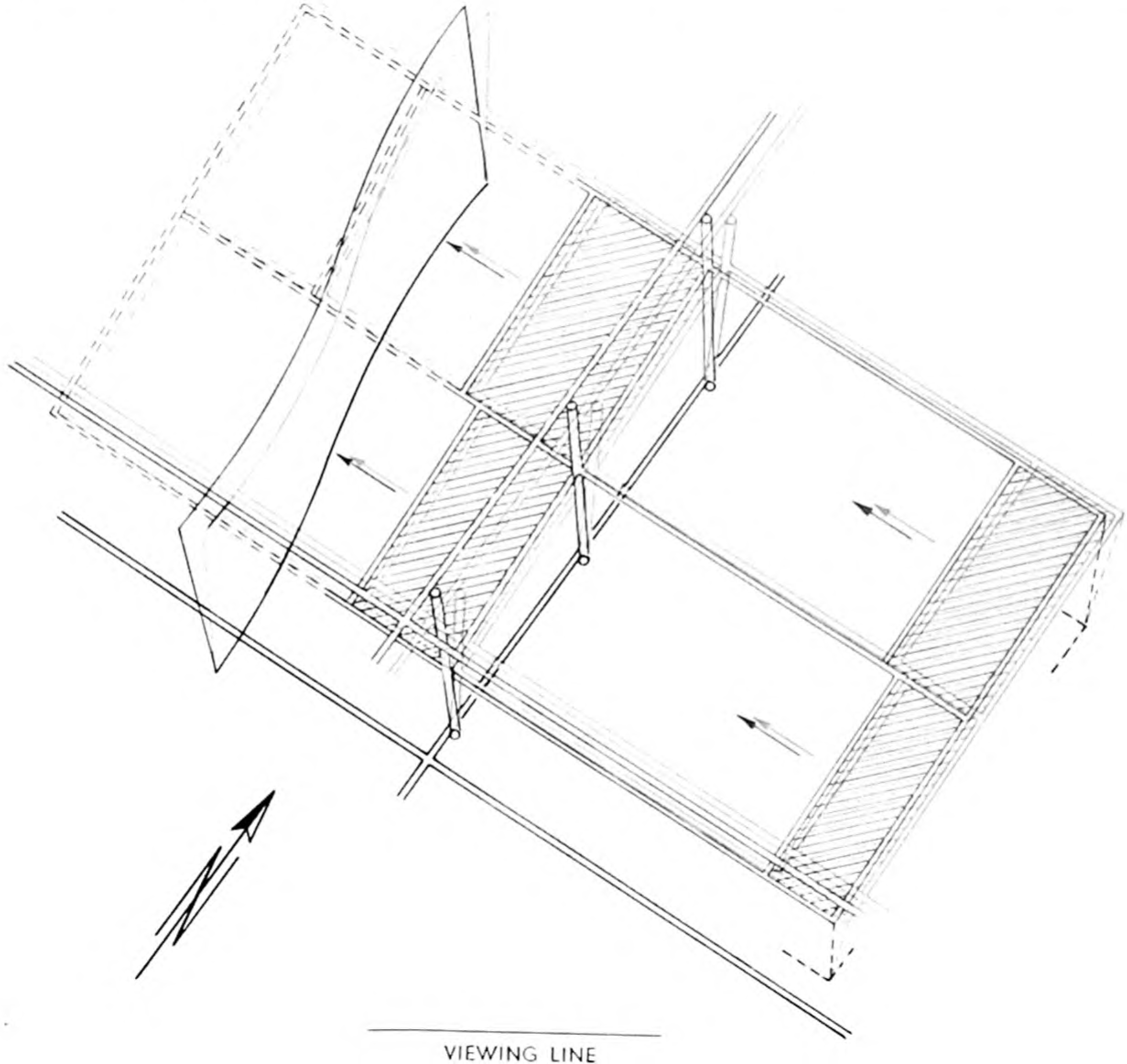


INDEX

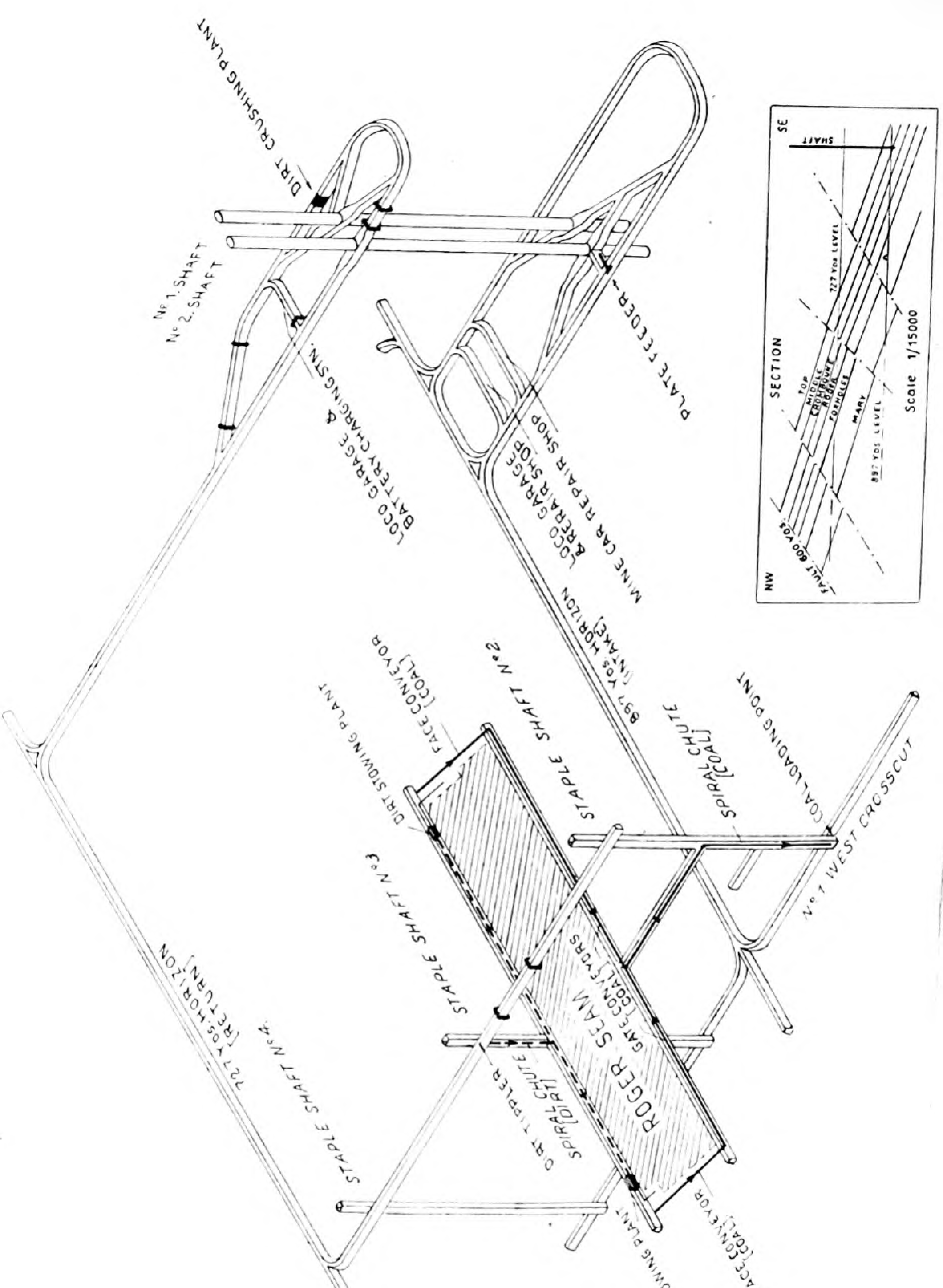
ANAGLYPHS OVERLEAF



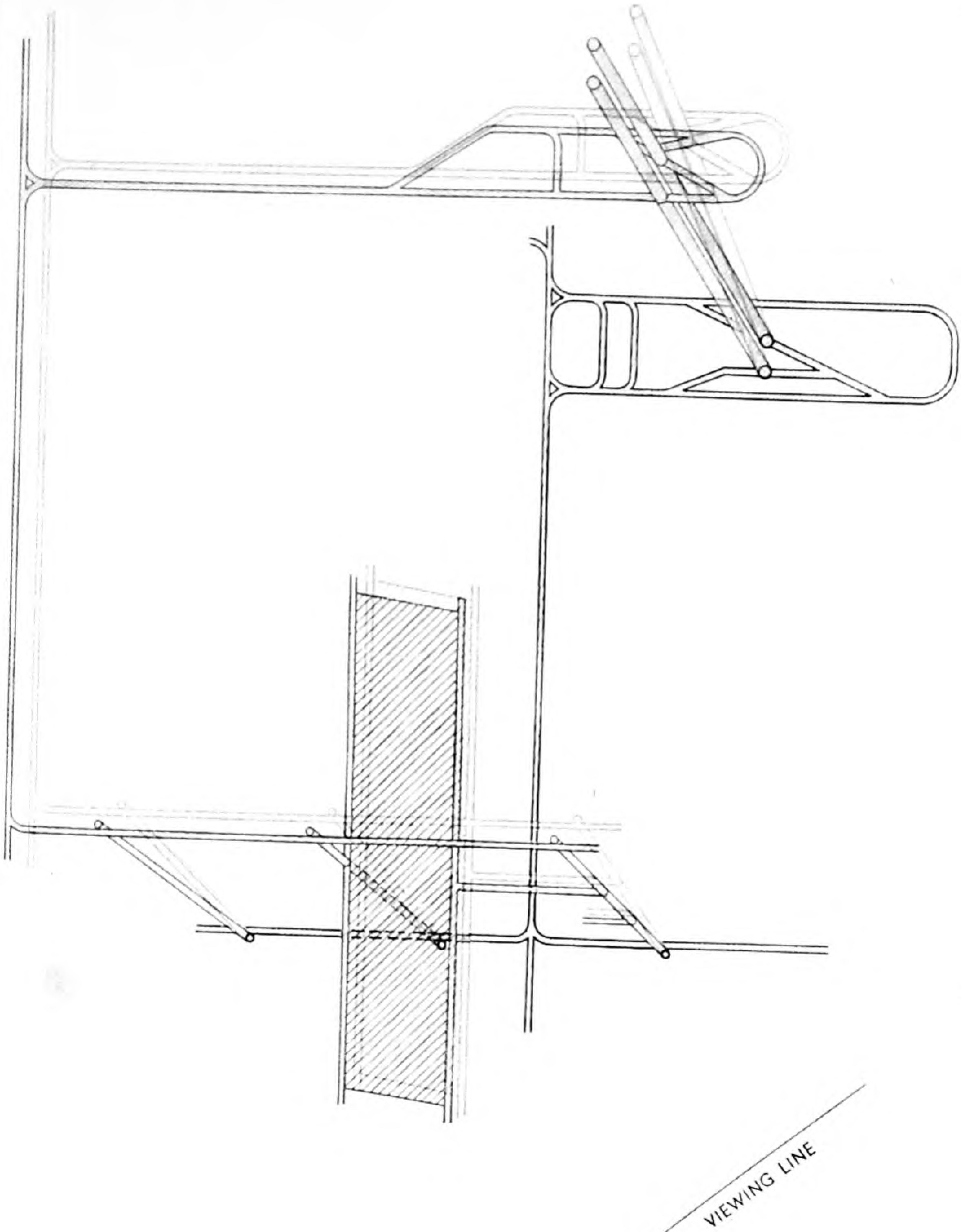
to upcast shaft
to downcast shaft
panel boundary
rise
f a u l t
face III
cross measure drift
main lateral 1st horizon
face I
rise
gate road 3
st. shaft 3
face IV
gate road 2
st. shaft 2
seam A
seam B
seam C
main lateral 2d horizon
st. shaft 1
gate road 1
face II
rise
1 in 22



VIEWING LINE



Shaft bottom layout and development at Bradford Colliery, Manchester.


VIEWING LINE